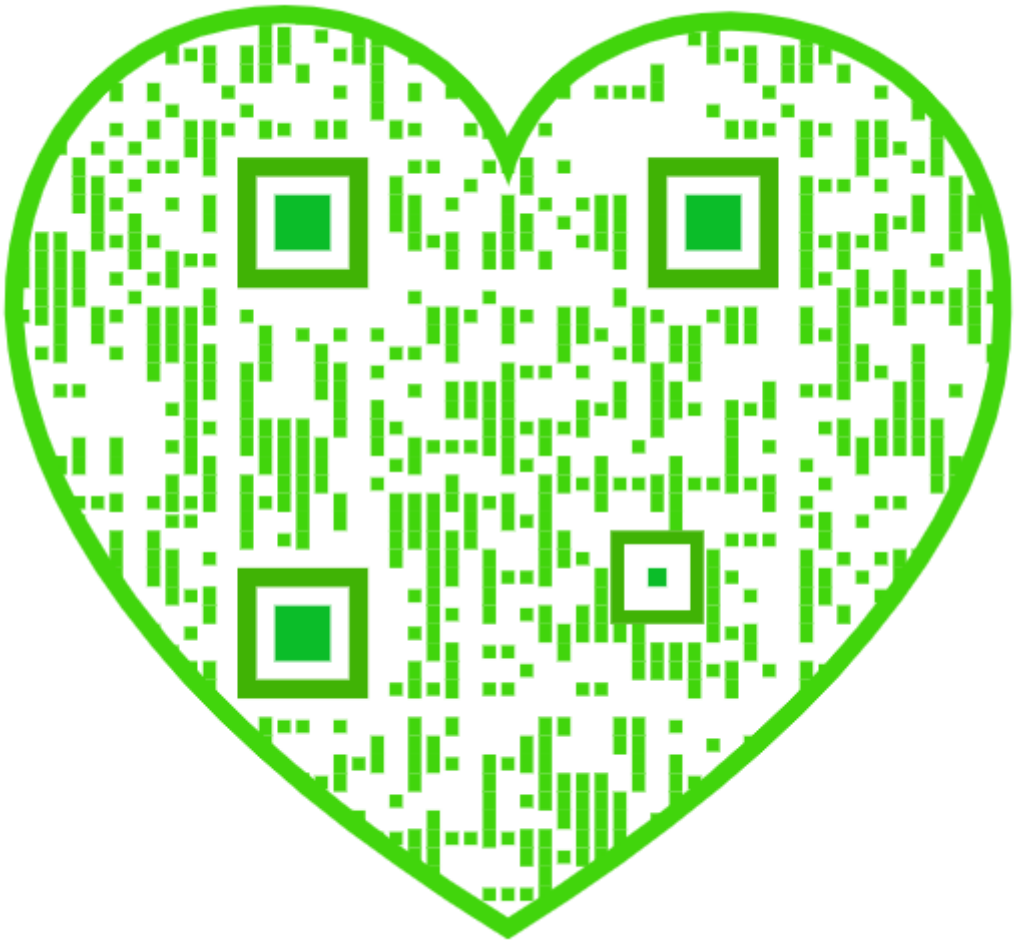

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THE PRACTICE OF COPPER SMELTING

BY

EDWARD DYER PETERS

Professor of Metallurgy, Harvard University; Author of "Principles of Copper Smelting," "Modern Copper Smelting"

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PREFACE

THIS work is written to replace my former book, "Modern Copper Smelting," and is intended to describe present metallurgical practice.

It consists, in the main, of two classes of material: Studies of actual operations and results at representative plants, supplemented largely by synopses of practising metallurgists; and investigations, comparisons, and discussions of these results, with a view to arriving at their fundamental principles.

These two threads are interwoven throughout the entire work, as they cannot be treated separately without sacrifice. The discussion of principles is, however, very brief, owing to the limitations of space, and this book must be regarded as a companion volume to my "Principles of Copper Smelting."

Owing to the great developments which the dry metallurgy of copper has experienced during the past twenty years, any attempt to write a complete work upon the subject would result in an encyclopedia rather than a handbook. I have, therefore, believed it best to introduce comparatively few illustrations of processes, but to select these from the best modern practice, and to study them in detail.

A book of this nature cannot be written without the coöperation of the owners and managers of a large number of metallurgical plants operating under a variety of conditions, and this assistance has been extended to me in the most liberal manner.

Among many sources of information it is proper to designate particularly The Anaconda Copper Mining Company, whose attitude toward the scientific investigator constitutes a personal favor to every student of metallurgy. Basing conclusions upon the treatment of unprecedentedly large quantities of material, handled and recorded in a thoroughly scientific manner, I am thus enabled to offer results that are both authoritative and invaluable. I desire especially to acknowledge the aid of the members of the staff of the Washoe Smelter: Messrs. E. P. Mathewson, William

Wraith, Frederick Laist, C. D. Demond, and W. M. Kelley, as well as of Foremen Estabrook, O'Brien, Dalton, Jetté, Beaton, Moran, and Keehew. I have also received great assistance from The Boston and Montana Consolidated Copper and Silver Mining Company through Messrs. C. W. Goodale, A. E. Wheeler, M. W. Krejci, W. T. Burns, and D. McKenzie, as well as from Foremen Elliott and Tracy.

I have received aid from Messrs. Hamilton Kilgour, J. C. Kilker, G. C. Riddell, H. J. Stehli, James Douglas, Thomas Kiddie, G. G. Vivian, Robert Sticht, H. B. Lowden, J. B. F. Herreshoff, Arthur L. Walker, Junjiro Kobuse, C. A. Heberlein, A. C. Clark, F. L. Antisell, and the late J. W. Nesmith and F. R. Carpenter. Many other acknowledgments appear in the text. Mr. W. S. Weeks, Instructor in Mining and Metallurgy, Harvard University, has aided me greatly in collecting and collating material.

EDWARD DYER PETERS.

DORCHESTER, MASSACHUSETTS, April, 1911.

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PRACTICE OF COPPER SMELTING

CHAPTER I

THE ORES OF COPPER

THE number of natural substances containing copper as an essential constituent is considerable, yet the list of minerals which are of sufficient commercial importance to be classed as ores of copper is short and simple.

For practical purposes, a copper ore may be defined as any mineral, or mixture of minerals, which contains sufficient copper to permit of its being worked to a possible profit.

METALLIC COPPER

Native copper is not uncommon as a product of decomposition in the upper portions of veins or deposits, but rarely persists to any considerable depth.

Its occurrence as a primary mineral is comparatively rare and seldom of commercial importance.

The most marked exception to this statement is the Lake Superior copper region, in northern Michigan, where it has been deposited in native form throughout a large area. It occurs extensively in the vesicular tops of ancient lava beds, the vesicles being filled with quartz, calcite, and other minerals, including metallic copper. Conglomerate beds, formed from fragments of the igneous rocks, also contain metallic copper, the most productive mine of the district — the Calumet and Hecla — being worked upon an extensive ore-shoot in one of the conglomerate beds.

CUPRITE; Red Oxide of Copper; Cu_2O

Cuprous oxide, containing 88.8 per cent copper.

This mineral occurs solely as a product of decomposition — commonly from sulphides, though occasionally from native copper. It is found, therefore, only in the upper, weathered portions of veins or deposits and, while seldom found pure in sufficient quantity to be of commercial importance, is occasionally disseminated through oxidized deposits so freely as to be of

decided value in raising the general tenor of the ore. It is softer than hematite; harder than cinnabar or proustite.

TENORITE; Black Oxide of Copper; CuO

Cupric oxide, containing 79.9 per cent copper.

This oxide occurs in various forms which are of interest to the mineralogist; but the miner is familiar only with the black, earthy, massive variety, which is termed Melaconite. This ore, usually resulting from the decomposition of sulphides, has been mined in considerable quantities in the Blue Ridge region of Tennessee, North Carolina, and Virginia, as well as in the Lake Superior district, but is rarely of much commercial importance.

Much of the so-called "black oxide" ore found in the zone of decomposition is really an earthy copper glance.

Two analyses, by Dr. Trippel, of the typical "black oxide" ore of the Blue Ridge region illustrate this point:

oxide of copper	5.75	3.8
ferric oxide	1.50	0.63
sulphur	18.75	25.40
copper (much metallic copper)	71.91	41.00
iron	0.93	26.56
soluble sulphates	0.72	1.78
	99.56	99.17

MALACHITE; $CuCO_3 \cdot Cu(OH)_2$

Basic cupric carbonate, containing 57.4 per cent copper.

Malachite is a bright green carbonate of copper formed by the decomposition of sulphide ores. While common in the weathered portions of deposits, and thus valuable as increasing the general tenor in copper, it is not often sufficiently massive to constitute an important ore in itself. Such exceptions, however, are not uncommon in North America in the arid regions of the south-west. Deposits of this mineral are frequently overvalued, on account of its brilliant color and its tendency to form thin incrustations over large areas of barren rock.

AZURITE; $2CuCO_3 \cdot Cu(OH)_2$

Basic cupric carbonate, containing 55.3 per cent copper.

This mineral is allied closely to malachite, but is much less common or important. When pure it has a light bright blue color, but is often dull blue and earthy. It is inclined to form

stains and incrustations and is, of course, a product of decomposition.

CHRYSOCOLLA; $CuSiO_3 + 2H_2O$

Cupric silicate, containing 36.1 per cent copper.

This green, or blue, copper silicate has frequently a characteristic, enamel-like luster, though it is sometimes earthy. It is a product of decomposition, and often forms crusts and concretions in the weathered zone of copper deposits. It seldom occurs in sufficient quantity to be of more than temporary importance, although in certain western arid districts of the United States there are quite extensive areas of low-grade chrysocolla which may be of value.

CHALCOPYRITE; Copper Pyrites; $Cu_2S \cdot Fe_2S_3$

Chemical composition: Cu, 34.5; Fe, 30.5; S, 35.

This is a brass-yellow mineral with strong metallic luster, often inclined to tarnish, and to acquire an iridescent coating of peacock colors. It is usually massive rather than crystalline, has a hardness of only 3.5 to 4, and a greenish black streak. Its softness and deeper yellow color distinguish it from pyrite, with which it is so generally associated. It weathers easily, forming the familiar gossan, or "iron hat," which so frequently distinguishes the outcrop of copper lodes.

Its distribution is almost universal. It is usually regarded as the "mother" mineral of the various copper ores; and it, together with cupriferous pyrite, is the original source of most of the other copper minerals. Its importance is well indicated in the following statement by Weed: "Chalcopyrite is, with cupriferous pyrite, the universal primary ore of copper deposits. It is found as a primary, original constituent of igneous rocks, both acid and basic; and, segregated, it sometimes forms with pyrite workable deposits in the basic rocks. It is also the chief copper mineral of contact metamorphic deposits, of complex copper-zinc-lead ores, and is found in many gold veins. It is the great universal copper mineral the world over. Though undoubtedly primary in many ores, yet it is also found as a secondary ore in many mines, and oftentimes an ore will contain both primary and secondary chalcopyrite."

In certain gold regions, chalcopyrite is highly auriferous, while in silver districts it is generally argentiferous.

COVELLITE; CuS

Cupric sulphide, containing 66.5 per cent copper.

This handsome blue mineral — often bluish black and earthy — had rarely appeared in sufficient quantity to be classed as an ore until the opening of the Butte district, where it occurs locally in considerable amount. It is manifestly a product of decomposition, being formed largely along rock fractures, and associated with kaolin.

CHALCOCITE; Copper Glance; Cu_2S

Cuprous sulphide, containing 79.8 per cent copper.

This is a blackish lead-gray mineral with streak of the same color, and inclined to tarnish to dull blue or green. Its softness is an aid in its determination. In the majority of instances where this ore occurs, it is found as a black, sooty, loose material at the junction of the upper, leached horizon of the vein with the deeper sulphide zone, and is of circumscribed extent. At Butte, Montana — and in certain other districts — it occurs in massive form, and extends to an unwonted depth, being still abundant 2500 feet from the surface.

Up to a comparatively recent date chalcocite was regarded as a mineral of secondary origin. At present, there is strong evidence that, in certain cases, it is a primary mineral. Whatever its origin, it is the most important copper ore of North America at present. Butte, Cananea, and much of Arizona base their copper production largely upon this mineral.

It is generally associated with pyrite and chalcopyrite, and may often be seen in process of weathering to cuprite and malachite. In silver districts it is likely to be argentiferous.

BORNITE; Erubescite; $3Cu_2S \cdot Fe_2S_3$

A variable copper-iron sulphide, the pure crystals containing Cu, 55.5; Fe, 16.4; S, 28.1.

The mineral goes under a variety of names: horse-flesh ore, purple copper, variegated copper, peacock copper, etc. It has a hardness of 3.5 to 4, and a specific gravity of 4.2. It has a brilliant metallic luster, and (properly) a brass-yellow color, which tarnishes through almost every shade of red, brown, blue, and purple. It often occurs massive, and may yield more copper than is called for by the above formula. This is usually due to the intimate admixture of chalcocite.

While bornite, in some rare instances, occurs under conditions which point to its being of primary origin, it is in the vast majority of cases a characteristic ore of secondary enrichment. It changes, on weathering, to iron oxide and copper glance, the latter mineral becoming, in time, copper oxide and carbonate. Bornite is one of the important ores of copper.

ENARGITE; $3Cu_2S \cdot As_2S_5$

Chemical composition: Cu, 48.3; S, 32.6; As, 19.1.

The development of large deposits of enargite in the deeper workings of certain of the Butte mines has now placed this comparatively scarce mineral among the *ores* of copper. It is an iron-black, massively crystalline mineral with well defined cleavage and bright luster, and has a certain resemblance to zinc blende.

In many of the copper mines of Chili, and sometimes in Colorado, this mineral was characteristic of the upper zones of secondary enrichment. At Butte it is not only a *deep* ore, but it occurs under conditions which indicate its primary origin.

Its high proportion of arsenic causes serious inconveniences at the smelter — due rather to the smoke difficulty than to its injurious effect upon the copper produced, as converting and electrolytic refining are competent to remove it from the metal.

Famatinite is a reddish gray mineral isomorphous with enargite, and similar in its composition, excepting that the arsenic is replaced by antimony. It is of some importance in certain of the copper mines of the Andes.

TETRAHEDRITE; Fahlore; $4Cu_2S \cdot Sb_2S_3$.

Gray copper ore; remarkably variable in composition; sometimes having a portion of its antimony replaced by arsenic, and thus graduating into tennantite. Much of the copper is replaced by iron, zinc, mercury, silver, lead, and other metals. Its copper content may vary between 25 and 40 per cent, and its silver values are frequently of especial importance.

It is a moderately soft, brittle mineral with metallic luster, and color ranging from iron-gray to steel-black. It cannot be classed as an important ore of copper, as it occurs usually in small quantities, and as a product of decomposition. On the whole, it is more important as an ore of silver than of copper.

CHALCANTHITE; $CuSO_4 + 5H_2O$

Natural bluestone, containing 25.6 per cent copper.

This is a hydrous cupric sulphate; translucent, bluish, and having the nauseating metallic taste characteristic of this class of salts.

It is deposited as incrustations or stalactites from mine waters which traverse the zone of decomposing copper sulphides. It rarely occurs in quantity sufficient to be mined as an ore, although in solution it may be valuable, metallic copper being precipitated therefrom by means of scrap iron.

A list of the ores of copper would not be complete without including the almost universal sulphide, *pyrite*, FeS_2, which, although containing no copper as an essential constituent, is in reality one of the most important sources of this metal. Apart from its almost constant occurrence (as cupriferous pyrite) in ordinary copper lodes, it forms the most extensive of all copper-bearing sulphide deposits, such as the great lenses of Rio Tinto, Mount Lyell, and the Rammelsberg. Much of the copper in the bodies of pyrite occurs in more or less concentrated patches or bands; usually as chalcopyrite, occasionally as one of the richer secondary sulphides. Much of the chalcopyrite in the upper portions of these deposits is evidently also of secondary origin, the values usually decreasing as depth is gained.

Where the pyrite is moderately rich in copper, it is not difficult to determine the presence of particles of chalcopyrite; but in the leaner ores it has not been practicable to determine whether the chalcopyrite molecule replaces a portion of the pyrite chemically, or whether it is merely present as a mechanical admixture. The weight of evidence appears to favor the latter supposition.

From the foregoing list it is obvious that the primary ores of copper are few in number, comparatively simple in composition, and — for the most part — widespread in occurrence. At the head of the list stands *chalcopyrite*, the almost universal primary mineral of copper veins or deposits, although also frequently found as a secondary mineral and important enriching agent to the upper zone of the original sulphides.

Enargite, although comparatively rare, is another mineral whose primary origin can scarcely be doubted in certain massive, deep-seated deposits.

Chalcocite and *bornite*, although in most instances manifestly secondary minerals, occur at times under conditions which indicate strongly their original deposition in these forms.

Apart from these few minerals, and from exceptional occurrences of the metal in native form, all the remaining copper minerals may be regarded as secondary compounds resulting from the oxidation, solution, and re-deposition of the primary copper sulphides.

The following cut [1] shows the various horizons which may be present in a copper vein that has undergone ordinary weathering, without too rapid denudation. It indicates the gossan zone at

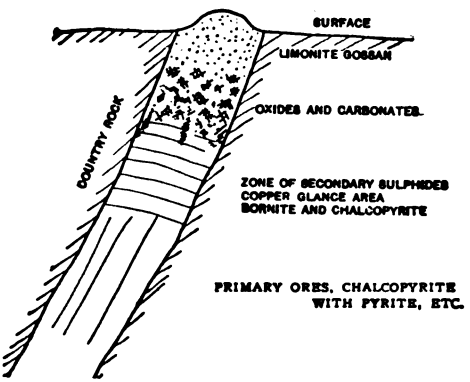


FIG. 1. — Weathering of ore in a vein

the surface; the zone of oxides and carbonates next below; the enriched sulphides beneath the oxides; and, finally, the unaltered chalcopyrite and pyrite (primary sulphides) which may be assumed to be the original filling of the vein.

[1] This cut is taken from "The Copper Mines of the World" by Walter Harvey Weed.

CHAPTER II

THE SAMPLING OF COPPER ORES

THE importance of possessing a tolerably exact knowledge of the value and constitution of all materials which are to be treated at the smelter is scarcely open to argument.

Where outside ores are purchased, their exact value *must* be determined as a matter of course, in order to arrive at the price which is to be paid for them. Where the smelter is supplied from its own mines, the exact determination of the metal values is not so vitally essential as in custom smelting, but is still necessary in order to check up the losses during treatment. Indeed, the only sound practice is to conduct the mine and the smelter as two separate industries, the mine selling its ores to the smelter on an established tariff, and each being thus responsible for its own expenses, extravagances, or misfortunes, and reaping the benefit of its own skill, economy, or good luck.

While the reasons just adduced refer mainly to the metal values of ores — the copper, gold, and silver — the technical metallurgist finds it equally important to pay attention to the non-metallic portions of his ores, such as silica, alumina, lime, iron, etc. He cannot recover his valuable metals unless he melts his entire mass of ore into a liquid, and he cannot fuse his ore into a liquid unless the substances just named are present in such proportion that they will melt together within the limit of temperature that he can obtain economically in his furnaces. Consequently, while the copper, gold, and silver are the sole substances contained in the ore which are important from a strictly commercial standpoint, the slag-forming constituents (silica, alumina, iron, lime, etc.) are the only vitally important ones from a strictly metallurgical point of view; but, as the copper, gold, and silver cannot be isolated unless the ore is first properly melted, it follows that — in conducting a smelter of any description — commercialism and metallurgical science go hand in hand,

and the slightest weakness in either branch will make itself manifest in the final result.

It is impossible for the metallurgist to mix his ores in such proportions that they shall yield a slag containing definite amounts of silica, iron, lime, etc., until he first knows just what amounts of these substances each individual parcel of ore contains. The science of chemistry offers rapid, accurate, and satisfactory methods for assaying and analyzing ores, and thus determining their values in gold, silver, and copper, as well as learning the exact percentage of silica, iron, lime, etc., which they may contain; but these methods are necessarily practised upon exceedingly small portions of the ore — seldom exceeding one ounce in weight. It is evident that it avails little to exercise skill and accuracy in determining the exact proportions of the constituents of this single ounce of ore unless that minute sample represents precisely the large parcel of ore from which it was taken, and which may weigh 100 tons, or even more. The process by which a small proportion is withdrawn from a large mass of ore, in such a manner that the small parcel shall represent, with reasonable accuracy, the entire heap of ore from which it was taken, is termed *sampling*.

This operation of sampling is an extremely difficult one to conduct with accuracy, and has exercised the best efforts of experienced men for many years to bring it even to its present stage of development. It is not yet entirely satisfactory, but must be described as it exists, some slight space also being devoted to its principles, imperfections, and possible chances for improvement, as indicated by those of the profession who have given particular attention to it.

Ore may come from the mine in all sizes, varying from lumps as large as a man can lift down to particles finer than flour. It is clear that no proper sample could be taken of fragments varying in size to such an extreme degree as this. The most obvious way to obtain an accurate sample from such ore would be to crush and grind the entire mass into fine particles and mix the same thoroughly, lessening the chances of error still further by drawing a large number of small samples from this thoroughly mixed and crushed ore, and mixing these again to form a final sample from which would be taken the small amount required for assay. This plan is impracticable for at least two reasons: first, the expense

would be prohibitive; second, the pulverized condition of the ore would interfere with the smelting process.

As in most technical operations, experience has established a compromise practice by which the more essential advantages of total crushing and mixing are retained to a considerable extent, while only a moderate proportion of the entire ore is crushed to a degree that is too expensive or too disadvantageous for the smelting process. This compromise practice consists in a series of graduated crushings, accompanied by withdrawals of lessening weights of sample-material as the ore is crushed finer. In this plan, the ore reserved for the sample diminishes greatly in weight with each crushing, and only a small proportion of the original ore ever arrives at a degree of fineness harmful to the smelting process.

Gold and silver ores contain small quantities of enormously valuable metals distributed, commonly in an irregular manner, throughout large quantities of comparatively barren gangue-rock. The high value and irregular distribution of these metals demand a care and expense in the sampling of their ores which is far beyond what is necessary for the lower values and greater dissemination characteristic of ores of copper. Even when the latter contain gold or silver, the precious metal is usually present in small quantity, and disseminated with the copper minerals throughout the gangue-rock. Hence, the sampling of copper ores is a decidedly cheaper, simpler, and less perfect operation than is the sampling of genuine gold or silver ores.

The steps involved in the attempt to withdraw a small, representative parcel from a large bulk of irregularly-sized ore are, first, to crush the entire lot of ore — or, at least, the entire amount of certain large units of the ore — to fragments of such size as experience has shown to be necessary for reasonably accurate results; second, to take from this entire lot of crushed ore a sufficient proportion of the same to be sure of representing properly the entire mass.

It is obvious that economy demands that the crushing of the entire mass of ore shall be carried no farther than is absolutely essential to secure accurate results, and also that the proportion withdrawn for the first cut of the sample — and which is to undergo successive crushings and handlings — shall be as small as possible. Experience is our most suitable guide on these

points, and the practice of the leading copper mines and smelters furnishes the proper basis for the brief directions to which a book of this character must confine itself.[1]

Examining, in the first place, the size to which it is necessary to crush the entire mass of ore which is to undergo sampling, it may be said — in general terms — that, under ordinary conditions, it ranges between two and one-half and four inches.[2]

Large jaw-crushers, or gyratory crushers, are used for this coarse crushing, and the original lots of ore must be of considerable magnitude to ensure reasonable accuracy. In practice, the individual lots of ore thus represented by a single sample usually run between 200 and 500 tons. If they are smaller, the first crushing should be proportionately finer.

From 15 to 20 per cent of the entire lot of ore thus crushed is cut out for the sample by methods to be described presently. The discarded portion may go at once to the ore-bins for treatment.

If the original lot of ore weighed 400 tons, and had been crushed to three and one-half inch size, twenty per cent might be taken for the first cut of the sample — thus weighing 80 tons. This might be crushed to one and three-quarters inches, and twenty per cent again taken for the second cut, weighing 16 tons. This again might be crushed to three-quarters of an inch for the third cut, and 15 per cent taken, weighing 2.4 tons, or 4800 pounds. Crushing this to one-quarter inch size, and taking 15 per cent, would give the fourth cut, weighing 720 pounds.

This final product of the main sampling mill would then be sent to the sample room, and be reduced to 50 pounds by means of the split-shovel. The 50 pounds would be ground in a

[1] Amongst many valuable contributions to the theory and practice of sampling, I refer to the following: "The Theory and Practice of Ore-Sampling," by D. W. Brunton, in *Transactions Am. Inst. Mining Engineers*, xxv, 826; "Modern Practice of Ore-Sampling," by Brunton, in *Mining and Scientific Press*, Oct. 30, 1909; "Principles of Ore-Sampling, by A. Van Zwaluwenburg, *Mines and Methods*, Oct., 1909; "Sampling by Machine," by T. R. Woodbridge, *Engineering and Mining Journal*, Nov. 7, 1908 and Jan. 30, 1909; "Causes of Variation in Ore-Sampling," by Thomas Kiddie, *Engineering and Mining Journal*, Oct. 23, 1909; "Conditions Necessary to Accurate Sampling by Machine," by John A. Church, *Engineering and Mining Journal*, July, 1908, and several valuable discussions of Mr. Church's paper by Philip Argall, E. P. Mathewson, L. S. Austin, and others, in later numbers of the *Engineering and Mining Journal*.

[2] One inch = 25.4 mm. One ton = 907 kilograms.

so-called coffee-mill grinder to pass a screen of 40 meshes to the linear inch, dried for several hours over steam-pipes, and reduced — by means of riffles — to about four pounds. This again would be ground to pass a 100-mesh sieve, mixed thoroughly in a mechanical mixer or by rolling on an oil-cloth, and divided into at least three portions — one for the smelter, one for the miner, and the third to remain sealed for the umpire's use in case of disagreement of results.

At certain works where the delivery of ore is large and regular, and the ore is low grade and tolerably uniform in character, only every fifth or tenth carload is subjected to crushing and sampling.

The proportion of each cut which it is necessary to withdraw, in order to obtain a representative sample, depends mainly upon the actual size of the largest fragments, assuming always that the total weight of the parcel of ore to be sampled is sufficiently large to ensure accurate results. On lots of copper ore less than 100 tons, it would be well to crush at least as fine as two and one-half inches, while on a 30-ton lot the first crushing should not be less than one and one-half inches unless the ore is of remarkably uniform tenor.

As both Argall and Brunton have pointed out emphatically, the size of the first cut, when the ore is in tolerably large fragments, should be very considerable, while the amount of the succeeding cuts may diminish materially, and still be as accurate as the larger first cut. Brunton has calculated the theoretical requirements of the successive cuts, and has shown how they may be diminished in quantity. In practice, there is not so general a diminution in the proportion of sample drawn from the successive cuts as would seem consistent with economy. It is, however, necessary to experiment in each individual case before one can establish the most advantageous compromise between expensive sampling with accurate results on the one hand, and cheap sampling with unreliable results on the other. Under ordinary conditions, the figures given in the preceding illustration indicate approximately the proportions of each cut which should be taken from the various ordinary sizes of fragments produced during graduated crushing and sampling.

We come, finally, to the methods and apparatus employed to cut out, or withdraw, the proportion of ore which is to be retained for the sample. I shall not discuss "grab-sampling," or trench-

ing, or similar crude devices which are occasionally adopted under stress of circumstances. These methods are too unsatisfactory and too obvious to require notice. The mere fact that their results may be profoundly influenced by the person conducting them is sufficient to condemn them completely in cases where opposing interests are at stake — as those of buyer and seller. They are, of course, less objectionable on fine than on coarse material, and on uniform than on "spotted" ores.

The methods in ordinary use for withdrawing the portion of the ore which is to constitute the sample are few in number. They may be classified as follows:

(*a*) *Hand methods of sampling.*
coneing and quartering
split-shovel
riffles

(*b*) *Machine methods of sampling.*

(*a*) *Hand methods of sampling.* — These are little used in America excepting for small lots of ores, or for completing the sample already taken from larger lots.

Coneing and quartering was the almost universal method of sampling ores until very recent times. As indicated by its name, the operation consists in piling a certain proportion (say every tenth shovelful) of the moderately crushed ore into a cone-shaped heap, taking care to deposit each shovel of material upon the apex of the cone in order that the fragments may roll down uniformly in every direction. Finally, the cone is flattened by means of shovels, or by a board, and is divided into quarters by right-angled lines passing through its center. The two diagonally-opposite quarters are discarded, the retained quadrants are shoveled into a new cone, and the operation continues — with occasional crushings of the ore — until only the few pounds remain which constitute the final sample.

This process requires far too much labor, time, and space to be in any way suited to modern conditions, and hence is not likely to be used on original parcels of ore, excepting for very small lots. As it may, however, be convenient at times, it will be proper to point out its most dangerous faults, as well as to indicate the measures by which these may be minimized.[1]

[1] I make free use of Brunton's excellent remarks on this method of sampling. *Mining and Scientific Press*, Oct. 30, 1909.

An inherent defect in this system lies in the fact that piling a lot of ore in the form of a cone does not mix it. On the contrary, it has a perceptible action in sorting the ore according to the size of its particles, the larger particles rolling downward and outward, while the smaller ones build up the center of the heap. This distinct separation of the ore according to the size of its particles would do no harm, providing the piling of the ore was conducted with such absolute precision and uniformity that each of the four quadrants contained exactly its proper complement of fine and coarse particles. This is, however, impossible, and, either through accident or design, the apex of the heap may be "drawn" several inches away from the proper center without

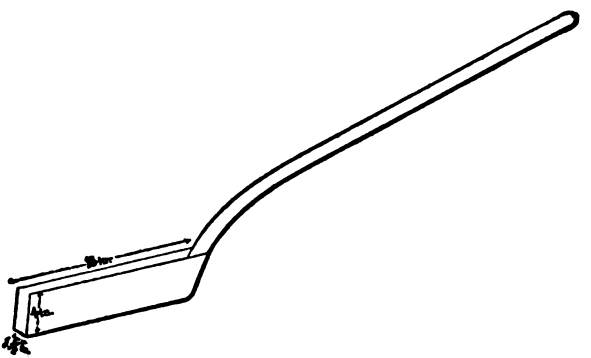


FIG. 2. — Split-shovel

detection by even the most careful observer. The cone may then be flattened out as usual, and divided into four perfect quadrants which will appear to be exactly uniform; yet one pair of the diagonally-opposed quarters will contain an excess of fine particles, and thus be unduly rich, while the other two quarters will suffer correspondingly. This form of inaccuracy may be obviated by coneing the heap about a fixed rod, or by the use of a divider consisting of a cross made of strips of sheet steel set on edge and crossing each other at right angles. The ore is thrown upon this divider while building the cone.

The *split-shovel* (Fig. 2) is used for small quantities of moderately fine ore, or for lessening the very considerable sample which is usually delivered to the sample-room from the main sampling-mill. It resembles a stable-fork with a long handle, the prongs

being replaced by several troughs made of sheet iron. The length of the troughs is usually about 16 inches, their width being about four times the diameter of the largest pieces of ore in the parcel undergoing sampling. One workman rests the split-shovel upon the floor, while a second man pours a thin stream of ore upon it from a square-pointed shovel. The portions of ore remaining in the troughs are retained, while those falling between are discarded. As the sample decreases in quantity, it must be crushed finer, and passed over a second split-shovel with still narrower troughs and spaces — say three-quarters of an inch in width. Its main use is for partly finishing a large sample which has already been reduced to moderate size — perhaps three-eighths of an inch.

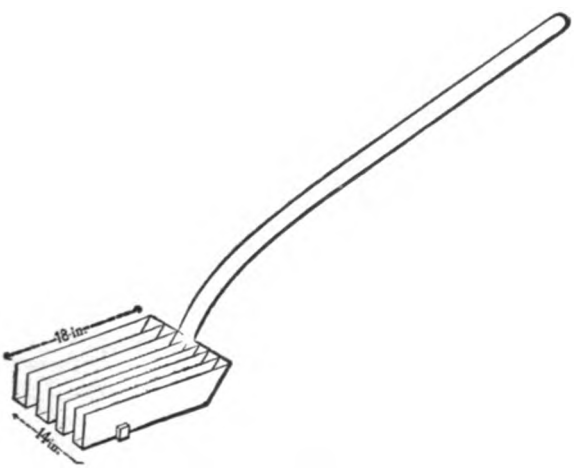


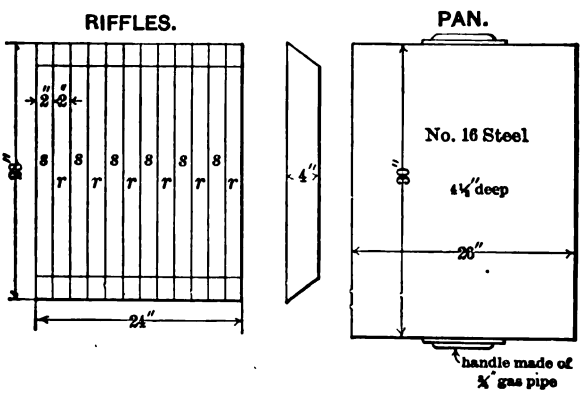
FIG. 3. — Split-shovel

This is, properly, a finishing device, and often used to reduce still further the diminished sample from the split-shovel. It consists of a series of parallel troughs, or riffles, alternating with corresponding open spaces, and formed of thin sheet steel. The riffles are commonly about three-fourths of an inch in width, and the whole rack fits into a receiving-pan for the sample.

An ingenious form of this device is known as the *Jones sampler*. It consists of a considerable number of inclined riffles resting upon a frame, and pointing alternately in opposite directions. It is used mainly for completing larger samples, or for sampling small quantities of fine ore. As with all devices of this nature,

it is necessary that the material to be sampled should be sufficiently fine.

(*b*) *Machine methods of sampling.* — The earlier attempts at mechanical sampling were not satisfactory, as they were based upon the continuous diversion of a certain fractional portion of a falling stream of ore. Experience has taught that no stream of ore is homogeneous either in its breadth or in its length, and that a machine designed to isolate only a fractional portion of such a stream will not yield a correct sample. The idea was then evolved of diverting the *entire* ore-stream for a *portion* of the time, and this principle forms the basis of the modern sampling machines. As well stated by Brunton:[1]



FIGS. 4 AND 5. — Riffle sampler

"It is not practicable to produce a stream of ore which shall be continuous in value through every part of its length any more than it is possible to produce a stream of ore that is constant in value throughout its width; but by taking a small sample entirely across a falling stream at short intervals, it is found that, while no single cut would give an exact representation of the composition of the entire lot, the average of thousands of these small samples is so nearly correct that results can be duplicated within very narrow margins; or, in other words, that individual errors are balanced.

[1] "Modern Practice of Ore-Sampling," by David W. Brunton, *Mining and Scientific Press*, Oct. 30, 1909. I make further reference to this article in succeeding paragraphs.

"The latest types of samplers are usually known as 'time-sampling machines' from the fact that they deflect the entire stream into the sample compartment for a varying portion of the time, depending upon the percentage of sample required. Treating the falling stream as a ribbon, they cut sample sections directly across its entire width, these portions varying in size and shape

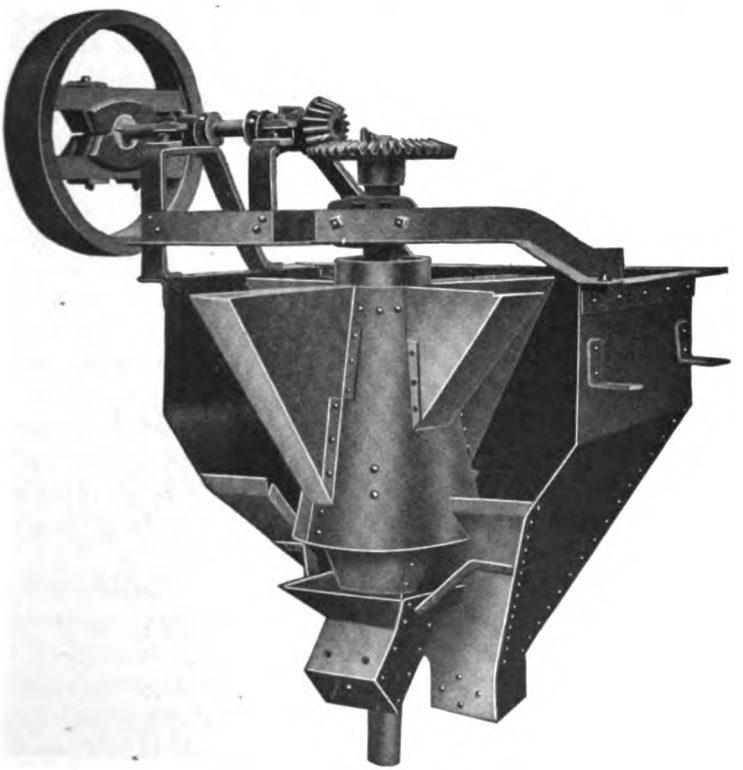


FIG. 6. — The Vezin automatic sampler

with the mechanism employed. Of the many types that have been invented and patented, three — the Charles Snyder, the Brunton, and the Vezin — have come into general use in this country. Both the Snyder and the Vezin samplers have sector-intake spouts revolving on a vertical axis, the difference between the two machines being that the delivery spout in the Snyder sampler is an annular quadrant, while the Vezin delivery pipe is either square or rectangular. The Brunton time-sampler

oscillates in a vertical plane through an arc of 120° instead of revolving in a horizontal plane like the sector-intake machines."[1]

The accompanying illustration of one of Taylor and Brunton's plants (Fig. 7) shows the general arrangement of a modern automatic sampling-mill with graduated crushing, equipped with four time-sampling machines, and arranged so that the final sample shall form sixteen-hundredths of one per cent of the original parcel or ore, or 3.2 pounds of sample for each ton of ore. The roll-feeders have been omitted to simplify the drawing.

Moisture-sample. — The determination of the moisture, even in an ore which appears quite dry, is an essential part of the sampling process, and has not as yet been established upon so satisfactory a basis as the ordinary sampling for metal values. As the small portion of pulp weighed out by the assayer for the determination of the metal values has had all of its moisture expelled by artificial means, it is plain that his results indicate the copper, silver, or gold which is contained in ore that is absolutely dry, and that, as the ore as delivered always contains a certain — constantly varying — proportion of moisture, it will be impossible to determine the weight of the perfectly dry ore unless the moisture is determined at the exact moment in which the ore is weighed. In practice it is impossible to fulfil such absolutely perfect conditions, but an endeavor is always made to approximate them.

Where the ore arrives in large shipments of tolerably uniform nature, it is common to determine the moisture by taking a grab-sample from each carload, rejecting the largest lumps. The sample may also be taken, at suitable intervals, from the ore-shoot leading to the bins or crushers. The weight of the sample thus taken is seldom more than one twenty-fifth of one per cent of the weight of the ore — one ton of ore, on this basis, being represented by eight-tenths of a pound of sample. This is crushed coarsely, and three or four pounds weighed out for the final sample, which is dried over steam-pipes for 12 hours or more, and weighed again when dry.

[1] As these time-sampling machines are standard articles, like steam-engines or blowers, it is not thought necessary to occupy space with extended descriptions and illustrations. Complete sampling-mills, adapted to every ordinary ore and to every reasonable condition, may be obtained from the manufacturers of mining machinery, together with expert assistance in planning and erecting it.

Where smaller lots of more valuable ore are in question, it is considered good practice to take the moisture-sample from the crushed and greatly-diminished material which results from the third or fourth cut of the time-samplers. This practice yields,

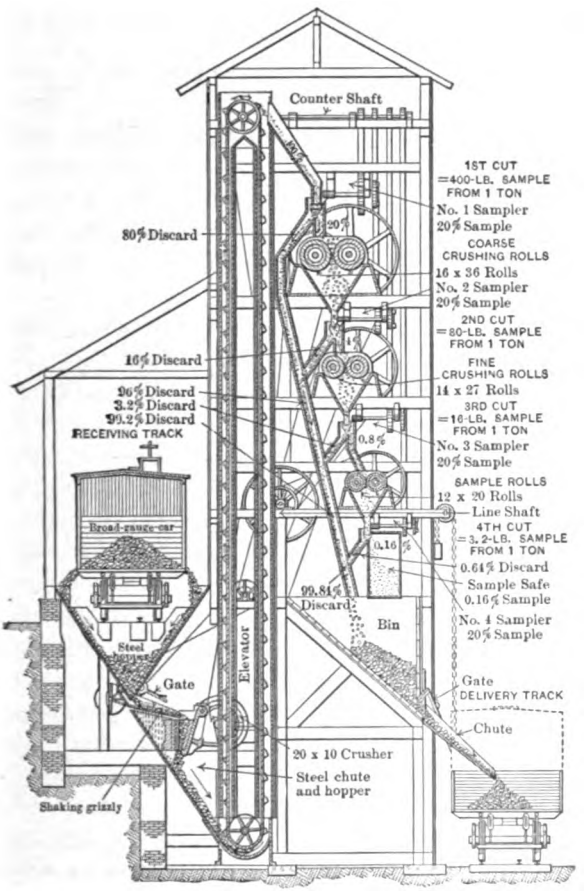


FIG. 7. — Sampling-mill

of course, a uniform and correct sample, but it is complicated seriously by the necessity of agreeing upon some "constant" which shall be added to the per cent of moisture thus determined, and which shall make up for the drying influences that the ore has undergone during the time intervening between weighing and sampling it, and, more especially, during its passage through the sampling-mill.

Exactly how large this "constant" shall be is a matter of agreement, as well as of disagreement, between miner and smelter. It is difficult to arrive at a figure which shall be mutually satisfactory; but, if once established, it saves much friction and inaccuracy.

An illustration of the method of conducting the sampling of the ores at a large smelter may be of interest. The Washoe Smelter of the Anaconda Copper Mining Company receives and smelts daily about ten thousand tons of copper ores from the Butte mines. The ore arrives in trains of fifty 50-ton cars, and is commonly sampled in the following manner, every car of first-class ore, and every fifth car of second-class ore, undergoing the sampling operation.[1]

The ore, in lots of 200 tons, is crushed to a size of three inches in the lowest level of the sampling-mill. It is then elevated to the top of the building, and begins its descent through a series of successive sampling and crushing operations. A Brunton time-sampler takes 20 per cent of its weight for the first cut (40 tons). This 40-ton lot is crushed to a size of one inch, and 20 per cent again taken for the second cut (8 tons). This is crushed by rolls to a three-quarter inch size, and 20 per cent taken for the third cut (1.6 tons). This is crushed by rolls to a one-quarter inch size, and 20 per cent again taken for the fourth cut, which thus weighs 0.32 tons, or 640 pounds. This corresponds to 32 pounds of sample for each ton of original ore, which is equivalent to sixteen-hundredths of one per cent. This 640 pounds is sent to the sample room and cut to 30 pounds with a split-shovel, and is then dried for 16 hours over steam-pipes containing steam at 16 pounds pressure. It is then ground in a coffee-mill grinder to pass a 40-mesh sieve, and is cut to three pounds with riffles. This final sample is ground to pass a 100-mesh sieve, mixed for 10 minutes in a mechanical mixer making 70 revolutions per minute, and is eventually put up in four separate envelopes.

The average moisture of the Anaconda Company's first-class ore for a period of six months was 2.56 per cent; second-class ore for the same period, 2.27 per cent.

[1] It is an interesting fact that the regular freight-rate on ores for the 30-mile haul from Butte to the smelter bins is twelve cents per ton.

CHAPTER III

METHODS OF COPPER EXTRACTION

COPPER may be extracted from its ores by a considerable variety of methods. The purpose of the present work is to study certain of the more important of these operations, and a convenient dividing line may be established by grouping all of the extraction processes under two heads: (*a*) wet methods; (*b*) dry methods.

(*a*) *Wet Methods.* — These processes are often designated as *chemical methods* because they possess so obvious a resemblance to the familiar laboratory operations of solution, precipitation, and filtration.

The main distinction between these wet methods and the more usual dry, or smelting, methods lies in the fact that in the wet methods the copper is dissolved by some appropriate liquid, while the great bulk of the ore remains comparatively unchanged.

The advantages of such a plan of treatment are obvious: for instance, if an ore contains two per cent copper and 98 per cent worthless gangue-rock, it would seem more reasonable to adopt a method of extraction which acts solely upon the 40 lb. of copper, rather than to melt the entire 2000 lb. of ore in order to obtain this small amount of metal.

Experience has taught us, however, that in the majority of cases it is more profitable to melt the entire 2000 lb. of ore than it is to attempt to extract the copper by a wet method. The reason for this conclusion is that, when we use these chemical methods on a large scale, we encounter a series of difficulties and hindrances which do not appear when these same processes are conducted in the laboratory. These difficulties are largely of a mechanical nature, and it is probable that they will be surmounted when the same combination of capital, energy, and skill is brought to bear upon them that is now exercised in the cyanide treatment of gold ores.

The mechanical aids to smelting have been so improved of late years that the mere melting of an ore is now comparatively a cheap operation, *provided the rocky portion of the ore is of such a nature that it will melt easily.* If, on the contrary, the rocky portion of the ore consists largely of substances which will not melt by themselves at a suitable temperature, and which therefore demand the addition of a considerable amount of limestone — or other barren fluxing material — the cost of smelting increases greatly, and often becomes prohibitive. In such cases the advantages of a wet method are obvious.

The wet processes for the extraction of copper are numerous, their chemistry is complicated, and their literature is extensive — though scattered. Any attempt to consider them in sufficient detail to be of practical value would increase the size and cost of this work out of all proportion to the value of the information given; and I feel that it is best to omit entirely this branch of metallurgy, and to confine myself to the dry, or smelting, processes.

There are large districts in the United States and Mexico — as well as in many other parts of the world — containing abundant copper ores of reasonable value, and free from the precious metals; the vein-rock is highly silicious; fluxing-ores are wanting; fuel and transportation are costly. These are the ores which are peculiarly fitted for wet treatment, and there is urgent need for a modern handbook devoted to this subject.[1]

[1] A mere compilation of existing literature upon the subject would be of little value. In order to satisfy present requirements, such a handbook must be written by a man thoroughly and practically familiar with the commercial smelting of copper ores, as well as with the cyaniding of gold. The former experience will fit him to weigh impartially the comparative advantages and drawbacks of wet or dry methods, and will enable him to escape the fatal influence of laboratory reactions; while the latter knowledge will aid him in the application of the splendid cyanide equipment to the treatment of copper ores. He must have had charge of a successful — by which I mean dividend-paying — wet copper plant for a considerable length of time, and should have been regularly engaged in the purchase and treatment of a variety of outside ores. If possible, he must not have invented a wet method of his own.

Such a man is at present not available; but when the great smelting companies find themselves overburdened with dry silicious ores, and become gradually forced into the manufacture of sulphuric acid, he will be forthcoming.

(*b*) *Dry Methods.* — The dry, or smelting, methods are all based upon one simple, fundamental principle: namely, that the valuable portion of the ore is heavier, bulk for bulk, than the worthless portion; thus, if we can succeed in transforming the entire mass of the ore into a fluid condition, so that the resulting liquid may have an opportunity to arrange itself in two layers according to weight, the heavy (valuable) portion will sink to the bottom, while the lighter (worthless) portion will float on top. When this result is attained, it is a simple matter to draw off the two layers separately, thus obtaining in a concentrated form the values that were formerly distributed through a much greater weight of material.

While many chemical changes take place during the actual melting of the ore, we may, for the moment, disregard these, and look upon the smelting process merely as a mechanical operation by which the heavy and the light portions of the ore are separated.

Silver and gold are often present in copper ores, and we cannot study the metallurgy of copper from its practical side without giving due consideration to these metals, which — although present in so small a quantity by weight as to exert no perceptible influence upon the smelting process — are often of the greatest importance commercially. Indeed, we frequently have ores carrying gold and silver which contain no other available metallic substances, and which would not yield their values if they were smelted by themselves. The minute particles of precious metal scattered through an overwhelming mass of gangue-rock have not body enough nor weight enough to sink through the melted rock and collect in a layer at the bottom. They need some bulky molten substance which shall drench through the mass like rain drops, dissolving and protecting each minute particle of precious metal, and carrying it down safely through the liquid contents of the furnace to form the heavy valuable layer at the bottom.

Such a substance is called a "collector" or "carrier," and must possess certain specific qualities: it must have a strong capacity for dissolving the gold and silver; it must be easy to separate from the precious metals; it must be easily procurable; it must have a sufficiently high specific gravity to separate thoroughly from the slag; it must, if possible, be valuable in itself.

There are only three substances that fulfil all of the requirements: namely, metallic lead, metallic copper, and copper matte.[1]

Metallic lead, resulting from smelting a mixture of lead ores with the ores containing the precious metals, is a most efficacious and satisfactory collector. Its use, however, is limited to districts where lead ores can be obtained at a reasonable price, and to smelting mixtures where there is not too much copper. Ores containing any considerable amount of copper are not smelted together with lead ores, as the mixed products of copper and lead that result from such an operation are expensive to refine.

This branch of the subject belongs to the metallurgy of lead and cannot be considered in the present work.

Metallic copper is an efficient collector for the precious metals, but is rarely produced in the smelting of ores. The reason for this will be considered later.

Copper matte is a most efficacious and satisfactory collector of either gold or silver. It is the almost universal collector in the smelting of copper ores, and as such will demand exhaustive study.

In my opinion, the study of copper smelting is simplified if we begin by regarding it purely as a mechanical operation by which we render an ore completely liquid, so that it may separate into two layers of differing specific gravity. We must also provide an opportunity for these two liquid layers to separate quietly and thoroughly.

In order to accomplish these purposes, we need some kind of a cavity, surrounded by walls which will not melt, in which to place the ore and to generate the heat required to melt it. Such a structure is called a smelting-furnace, and any advantage which one type of smelting-furnace may possess over another type results merely from its cheapness, convenience, suitability of form, perfection of mechanical details, or similar attributes. Consequently, the statements so frequently made by inventors or manufacturers of "patent smelting furnaces" that their apparatus will melt the most refractory ores without flux, or similar claims in defiance of the laws of nature, are absolutely false.

[1] In rare instances an iron matte, free from copper, is used as a collector of the precious metals, and even the arsenides and antimonides of certain metals have been similarly employed. These unimportant exceptions will be considered in a later chapter.

The one essential and fundamental operation which ores of copper must undergo when treated by a dry method is *smelting;* and, having reduced the smelting furnace to a simple cavity surrounded by infusible walls in which we can generate the degree of heat required to melt a suitable ore, we may proceed to reduce to the lowest terms possible the smelting operation itself.

For this purpose it is helpful to regard every ore as consisting of two distinct portions: a metallic portion and an earthy portion.

The metallic portion would, of course, comprise any metals which might be present as *metals*, and would also include the oxides of such metals as are changed into the metallic condition by smelting. These substances, however, are usually confined to surface ores which are exceptional, and soon exhausted; so that the metallic portion with which the ordinary smelter has to deal consists mainly of the sulphides of iron and copper, accompanied frequently by a small quantity of the sulphides of other metals: galena, zinc blende, etc.

These iron and copper sulphides — as well as most of the accompanying metallic sulphides — melt at a moderate temperature, and mix together to form the heavier of the two molten layers which result from the fusion of the ore. This new sulphide mixture is called *matte*, and, under ordinary conditions, may weigh something like three-fourths as much as did the original sulphides that were contained in the ore from which this matte was formed.[1]

This insignificant loss of weight which the sulphide portion of the ore undergoes, when melted without excess of air, results mainly from the driving off — by heat alone — of a fractional part of the sulphur from certain of the sulphides. It follows, therefore, that the behavior of the commoner sulphides when exposed to heat — both with and without the free access of air — is a point of fundamental importance to the metallurgist.[2]

An important commercial deduction may be made at once

[1] By "ordinary conditions" I mean conditions in which no excess of air is admitted into the furnace beyond the amount that is necessary to burn the coal or coke which is required to melt the ore. This assumption simplifies the study of the operation of smelting at the outset, and will be modified later to correspond with modern practice.

[2] This point is discussed in detail in Chapter III.

from this behavior of the sulphides: namely, if sulphides lose only about 25 per cent of their weight in smelting (without excess air), there is little advantage in smelting a massive sulphide ore by itself (without excess air).

The main object of smelting an ore is to remove a considerable portion of its worthless material, and to obtain its valuable constituents in a small weight of concentrated product upon which we can afford to lavish considerable expense for separating and refining. This concentrated product will consist mainly of the sulphides of iron and copper, for the simple reason that these are the predominating sulphides in ordinary copper ores. It will also contain the greater part of the incidental sulphides (arsenides, antimonides) already mentioned; and, if the ore contained gold or silver, these metals will also be dissolved in this new artificial sulphide (matte).

It will then be clear that the main effort in smelting an ordinary ore of copper is not directed towards the sulphides which it may contain, because these sulphides undergo but little change by mere melting. The chief purpose is to separate the metallic portion of the ore from the earthy portion; and, as the earthy portion usually is present in much greater quantity than the metallic part (sulphides), it is evident that a normal smelting operation will yield two products:

1. A small, rich, metallic portion — matte.
2. A large, worthless, earthy portion — slag.

The earthy portion of the ore comprises all of the constituents which were not included in the metallic portion, and consists mainly of oxides, such as oxide of silicon, or silica (SiO_2); oxides of iron; limestone, which, when heated, loses its CO_2 as a gas, and becomes oxide of calcium (CaO); oxide of magnesium (MgO); oxide of aluminum (Al_2O_3); and various other oxidized substances of lesser importance.

As has been already indicated, the aim of the smelter is to melt together all of these earthy substances into a liquid, so that the molten metallic portion of the ore may separate from it with perfect freedom, and may settle to the bottom of the furnace. The liquid resulting from the melting together of the earthy constituents is called *slag*, and is the chief object of the smelter's solicitude. The reason why the metallurgist bestows most of his attention upon his worthless slag, rather than upon his valu-

able matte, becomes apparent as soon as we understand the nature of these two products.

In the first place, the metallic, or matte-forming, constituents melt and mix together in any and all proportions, while the earthy, or slag-forming, substances cannot melt into a liquid slag unless their respective amounts are proportioned within quite narrow limits.

Again, the matte-forming constituents melt at a temperature considerably below that required to melt the slag-forming substances. Consequently, the metallurgist knows that if he provides a temperature sufficient to melt the earthy constituents of an ore into slag, the metallic portions of the same ore will melt into matte without the slightest difficulty or attention.

Thus, as the greater includes the lesser, the production of a suitable slag includes the production and separation of an appropriate matte — always provided that the work is executed under proper conditions.

We may then in this brief introductory consideration of the smelting process assume that we already understand something of the laws governing the formation of matte; and, having thus eliminated the matte-forming constituents of our ore, we may devote our entire attention to the much more difficult slag-forming (earthy) substances.

The main reason why it is so much easier to melt the metallic constituents into a matte than it is to melt the earthy constituents into a slag arises from the fact that (with unimportant exceptions) each of the various metallic sulphides melts at a comparatively low temperature (925° C., or less), and any or all of them, when thus melted, mix together to form a fluid matte.

The earthy substances, however, behave in a manner that is almost precisely the reverse. For the sake of simplicity, I will confine this preliminary illustration to the three slag-forming substances which are the most common and infinitely the most important: silica, iron oxide, and calcium oxide. Taken alone, neither silica nor calcium oxide melts at any temperature within the range of feasible metallurgical operations; even the iron oxides require an unduly high temperature, and yield a heavy, sticky, unsuitable slag.

If, however, these three substances are mixed together in suitable proportion, they yield a perfect slag: fusible at a mod-

erate temperature; having a moderate specific gravity and a proper fluidity, so that the matte globules can settle out of it completely; and possessing pretty much all of the qualities which it is reasonable to demand.

It is plain, therefore, that before a man can undertake the management of even the crudest smelting operation with any hope of success, he must know what proportions of these earthy substances are necessary for the formation of a suitable slag; while, if he is to engage in any smelting venture on a considerable scale, or where he is exposed to competition, he must have an intimate knowledge of slags.

For the moment, I will point out the fundamental principles of slag-formation, referring the reader to a later chapter for a more thorough — though still only partial — consideration of the subject.

Up to this point, we have been considering smelting purely as a *mechanical* operation, by which the ore is afforded an opportunity to separate into two layers of differing weights. We come now to the point where *chemistry* assumes the principal part.[1]

As copper slags are always silicates, there is one substance — and only one — which is a universal and essential constituent; this is silica, SiO_2. While slags are usually mixtures, or mutual solutions, of two or more silicates, it will be convenient for present purposes to regard each individual slag as a simple salt, consisting of silica on one side and various bases on the other side: silica + bases. Thus we speak of the silica side (or acid side) of a slag, and of the base side.

Consequently, on this assumption, if we determine in the laboratory the percentage of silica which any given slag contains, we also know at once its content in bases; for all that is not silica is assumed to be base.[2]

According to this assumption, then, if a slag contains 40 per cent silica, it must also contain (100 − 40) 60 per cent base.

The silica content of ordinary copper slag ranges from 30 to

[1] I shall endeavor to explain the important subject of slag-formation in a manner that will be intelligible to readers who have but little knowledge of chemistry.

[2] It is better to begin the study of slags with a simple assumption of this nature than to present it obscured by exceptions, or by attempts at extreme accuracy of statement.

45 per cent. Slags outside of these two extremes are exceptional, and demand unusual skill on the part of the metallurgist.

We have thus already simplified our introductory consideration of ordinary slags by learning that they must contain from 30 to 45 per cent silica and, consequently, from 70 to 55 per cent bases. As it is inconvenient to employ such variable figures, I will for the moment assume that we intend to produce a slag containing 40 per cent silica, and we may express its analysis as follows:

silica	40	per	cent
bases	60	"	"
	100	"	"

As each 40 lb. silica requires 60 lb. bases to form the above slag, 1 lb. silica will require $(\frac{60}{40}) = 1.5$ lb. bases. This gives us a factor by which we can tell at once from the analysis of an ore whether, when smelted, it will yield a slag exactly of the required grade; whether it contains too much or too little silica; and, if so, how much of either substance must be added to the furnace-mixture to produce a slag containing exactly 40 per cent silica and 60 per cent bases.

The following problem, which is an elementary type of what is constantly occurring in practice, will illustrate the simplicity of the calculation for determining whether a given ore contains too much or too little silica to produce a slag of 40 per cent silica and 60 per cent base; or how much of one or the other of these substances must be added to it to effect the desired result.

In order to make such a calculation we need to know: (*a*) How much base 1 lb. silica requires to yield a slag containing 40 per cent silica. (*b*) The composition of the ore, to show its content of slag-forming constituents.

We have already demonstrated that 1 lb. silica requires 1.5 lb. base to yield a slag containing 40 silica and 60 base. A partial analysis of the ore shows that it contains slag-forming constituents as follows:

silica	46	per	cent
bases	33	"	"
	79	"	"

Employing as a unit of weight the ordinary American ton of

2000 lb., it is evident that each ton of the above ore contains

$$2000 \times 0.46 = 920 \text{ lb. silica.}$$

As each pound silica requires 1.5 lb. base to produce the desired slag, the 920 lb. silica requires $920 \times 1.5 = 1380$ lb. base.

The ore contains already $2000 \times 0.33 = 660$ lb. base, so that there is lacking $1380 - 660 = 720$ lb. base, which must be added to the furnace-charge before the mixture can produce the desired slag of 40 per cent silica and 60 per cent base.

Thus the final ore-mixture will consist of one ton of the original ore plus 720 lb. of added bases, and must contain the following slag-forming constituents:

silica	920 lb.
bases	1380 "
Total	2300 "

This mixture, therefore, contains

$$\frac{920}{2300} = 40 \text{ per cent silica}$$

$$\frac{1380}{2300} = \frac{60}{100} \text{ per cent bases}$$

It will be noticed that I have said nothing about the *quality* of the different bases which make up the 1380 lb. of basic material in the preceding illustration. So far as my statements have yet proceeded, this 1380 lb. may be made up of any one or two or all of the basic substances which ores or fluxes may contain, such as the oxides of iron, calcium, magnesium, aluminum, barium, manganese, sodium, potassium, etc. I have, for the sake of simplicity, lumped all of these basic substances together, and called them "base"; and I have assumed that any slag will be satisfactory so long as it contains 60 per cent base and 40 per cent silica. As I have already mentioned, this assumption is absolutely incorrect; but it simplifies materially the study of the quantitative requirements of a slag, and when this point is once understood, the consideration of its qualitative composition is easy to follow.

Reference to the preceding calculation shows that each ton of our assumed ore must be supplied with 720 lb. of base in order

to yield the required slag. The manner in which this base is to be procured introduces at once a commercial question; and, as the object of mining and smelting is to make a profit, it is proper that our chemical remedies for difficult situations should be in harmony with business principles.

From a strictly chemical standpoint we have completed our problem and attained our purpose when we have demonstrated that each ton of the ore at present under consideration will require the addition of 720 lb. of base to yield a suitable slag on smelting. Nothing more need be done except to mine or purchase a sufficient supply of basic flux to fulfil this requirement.

Let us now view the same illustration from the standpoint of the commercial smelter, and learn how serious a matter it may be to undertake to treat an ore which carries a considerable excess of silica.

Assume a single blast furnace, having a daily capacity of 200 tons of charge (ore plus flux), and intended to smelt the ore used in the preceding illustration. Let us see, roughly, what it will cost to flux the excess silica under these conditions.

Calcium oxide (lime) is ordinarily the commonest and cheapest of the bases. We have learned already that one ton of this ore will require 720 lb. of base; but this statement does not mean that we have only to add 720 lb. of limestone to each ton of ore. An absolutely pure limestone consists solely of calcium, oxygen, and carbon. After being submitted to the heat of the furnace, all of the carbon and much of the oxygen escapes from the chimney as carbon dioxide (CO_2), leaving its calcium combined with the rest of the oxygen as calcium oxide (CaO). This is the base which we desire, and which is familiarly known as lime. Ordinary limestone may yield about one-half of its weight as lime available as a base for fluxing purposes. Consequently, instead of adding 720 lb. flux to each ton of ore, we are obliged to add $720 \times 2 = 1440$ limestone, in order to furnish the 720 lb. CaO.

We have now the data necessary to calculate the daily useful capacity of our furnace. Our smelting mixture, based upon one ton of ore as a unit, consists of

ore	2000 lb.
limestone	1440 "
Total	3440 "

or, expressed in percentages in round numbers,

ore	= 58	per cent
limestone	= 42	" "
	100	" "

It is obvious then that, apart from the considerable first cost of 42 tons of limestone, and of certain other serious disadvantages which will be considered later, no less than 42 per cent of the entire capacity of our plant and of our fuel will be occupied in smelting barren limestone.

Something like these imaginary conditions are present in the great majority of cases, and there are three remedies for them which are so familiar and so obvious that the mind of both the miner and the smelter passes them in review almost automatically when confronted with such a situation. These are

(*a*) Mechanical concentration of the ore.

(*b*) Purchase of basic ores to act as flux for the excess silica.

(*c*) Modification of the original ore in such a manner that it may furnish its own base.

(*a*) *Mechanical concentration of the ore.* — This familiar operation, by which the light, earthy portions of the ore are washed away, leaving the heavy, metallic portion in a much smaller bulk of so-called "concentrates" is, of course, the most simple, obvious, and advantageous plan that can be adopted, provided the nature of the ore admits of using it. It is peculiarly advantageous in that it lessens the amount of material to be treated by the more costly smelting operation. It will be noticed that, mechanically, it is based upon the same principles as smelting. In both cases the object of the process is to place each particle of the materials forming the ore in such a state of freedom that it may assert its specific gravity and become either a part of the heavy, valuable, metallic layer at the bottom, or of the light, worthless, earthy layer on top.

As a general proposition, an operation increases in expense of running, as well as in cost of plant, somewhat in proportion to the temperature at which it must be conducted; and it is a much more costly and critical operation to conduct the smelting of an ore at a temperature of 1400° than it is to execute its mechanical concentration by water at the temperature of the air.

Therefore, when it is feasible, we are always glad to substitute

the cheap process of concentrating by water for the expensive concentrating by smelting; but, as is known to all, there are many ores — especially those of copper and silver — which are so friable and brittle that they crush into fine slimes and undergo excessive loss of metal values.

(*b*) *Purchase of basic ores as a flux for the excess silica.* — Certain ores carry iron, lime, or other bases in proportions greater than are required to flux their own silica: that is to say, they contain more than 1.5 lb. available base to one pound silica. This is often called *excess* iron, or *excess* lime, and furnishes a welcome flux for the superfluous silica of other ores.

Where such basic ores can be mined or purchased on profitable terms, they are peculiarly advantageous to the smelter; for they not only furnish him his basic flux, but also yield a profit from their own treatment. Thus his entire furnace capacity is employed upon profit-making material; and, instead of smelting only 116 tons ore (and 84 tons flux) in his 200-ton furnace, he can now smelt 200 tons ore. It costs as much per ton to smelt worthless flux as it does to smelt profitable ore. It is clear, also, that the entire cost of running the plant has got to come upon the *ore* which is smelted — be it much or little.

Consequently, on an assumption that it costs the metallurgist $600 per day to run his 200-ton furnace, his cost of smelting per ton of ore, besides the cost of the limestone, would be, when using 84 tons limestone as flux,

$$\frac{600}{116} = \$5.17$$

when using basic ore as flux,

$$\frac{600}{200} = \$3.00$$

This difference per ton in the cost of smelting ore would in itself constitute a satisfactory margin of profit at almost any large smelter.

Of course it will usually require a much greater weight of basic ore than of limestone to furnish a given weight of excess base. I am assuming, however, that basic ores are abundant, and can be obtained on a reasonable margin of profit. (Both of these conditions are unusual.)

This plan is evidently a highly advantageous one, and is

used at almost all smelters to a greater or less extent. There are two reasons, however, why its application is usually limited. In the first place, a silicious district seldom affords any adequate supply of basic ores, quartz being so universal a gangue mineral. Again, these basic ores are so desirable to almost every smelter, that competition for them becomes very keen, and they often command so high a price that their employment offers little advantage above that of barren limestone.[1]

(*c*) *Modification of the original ore in such a manner that it may furnish its own base.* — In order to understand why I have included this third plan among the remedies for an excess of silica, it will be necessary to refer again to my division of the constituents of every ore into two portions — metallic and earthy. The metallic portion consists of the sulphides of metals (pyrite usually predominating greatly); and, as it simply melts into a matte and separates at once from the earthy portion, it may be regarded as neutral — so far as the slag is concerned.

The earthy portion is made up of the slag-forming substances and may, of course, consist either of an acid earthy part (silica), or of a basic earthy part (bases), or of a mixture of both. As already indicated, there is usually far too much silica to form a proper slag with the bases present (40 silica to 60 bases). Consequently, the acid earth generally predominates, and we need a considerable addition of basic earths to produce the desired slag. Thus far we have considered two methods for the correction of this excess silica. We might (*a*) crush the ore and wash away the silica (mechanical concentration); or, (*b*) add bases from outside sources until we arrive at the desired ratio of one and one-half base to one acid.

[1] I feel justified in emphasizing this point particularly as, in my opinion, the failure of new smelting enterprises results more frequently from an excess of silica in the furnace-mixture than from all other causes combined. During the past 40 years I have witnessed the failure of 14 smelting plants with whose circumstances I was more or less intimately acquainted. In 12 of these instances, over-silicious ores and lack of economical basic flux was the direct cause of their ruin. The two other cases were complicated by financial embarrassments, resulting — in the one instance — from dabbling in patent furnaces which could smelt *any* ore, no matter how silicious; in the other case — from becoming involved in railway building to tap a district which could furnish basic ores to flux the excessive proportion of silica which their own ore contained.

The method now under consideration is to modify the ore itself in such a manner that a part of its metallic (neutral) constituents shall be converted into basic substances, and thus become available to flux the excess acid portion of the ore. This is rendered feasible by the fact that one important and plentiful constituent of many ores may belong either to the metallic portion or to the basic earthy portion, according to its chemical condition. This substance is iron, which, when combined with sulphur, forms metallic compounds and enters the matte; but, when freed from sulphur and combined with oxygen, forms basic earthy compounds, and thus enters the slag — constituting our most common and valued flux for silica.

The commonest method at our disposition for changing the neutral metallic iron sulphides into earthy, basic iron oxides is the familiar operation of roasting — a process so common and so simple that its extraordinarily advantageous effect is not always fully appreciated by the non-technical investor. Its value becomes most strikingly apparent, of course, in the case of ores consisting solely of quartz and metallic sulphides, thus containing no basic earths whatsoever.

Disregarding, for the moment, any copper sulphides which may be present, let us imagine an ore consisting exclusively of quartz and pyrite, and containing per hundred pounds

metallic portion — pyrite	75 lb.
earthy portion — quartz	25 "
	100

or, arranging its constituents in a more graphic manner,

metallic portion — pyrite			75 lb.
earthy portion	acid earths — quartz	25 lb.	
	basic earths	00	25
Total			100 lb.

This ore would of course be unfit for smelting as it now stands; for the pyrite would fuse into a matte, while the 25 lb. of quartz — having no earthy bases with which to combine — would remain unmelted. Let us now examine the effect produced by roasting this same hundred pounds of ore.

If it were thoroughly roasted until all of the iron sulphide (FeS_2) was converted into iron oxide (Fe_2O_3) — which in actual practice is neither feasible nor desirable — the end result would

be as follows: one hundred pounds of the raw ore would yield after roasting

metallic portion		00 lb.
earthy portion	acid earths — quartz	25 "
	basic earths — ferric oxide	50 "
Total		75 lb.

There is yet one further complication to remove before we can determine the composition of the slag which will result from smelting the roasted ore. Ferric oxide (Fe_2O_3) is reduced during the smelting operation to ferrous oxide (FeO), and enters the slag in the latter form.

Fifty pounds ferric oxide will yield 45 lb. ferrous oxide. Consequently, the slag-forming constituents of the 75 lb. of roasted ore (resulting from 100 lb. raw ore) will consist[1] of

metallic portion		00 lb.
earthy portion	acid earths — quartz	25 "
	basic earths — ferrous oxide	45 "
Total		70 lb.

This, if melted, would yield a slag consisting of 35.7 per cent SiO_2 and 64.3 per cent FeO, thus actually containing more base than we need in order to produce the slag which was adopted

[1] The details of this calculation are too important to be omitted from an elementary consideration of the subject, and are as follows:

Pyrite (FeS_2) consists of one atom Fe and two atoms S. Atomic weight of Fe = 56. Atomic weight of S = 32.

1 atom Fe = 56 = $\frac{56}{120}$ = 46.7 per cent
2 atoms S = 64 = $\frac{64}{120}$ = 53.3 " "
120 100

Thus pyrite contains 46.7 per cent iron; and the 75 lb. pyrite of the 100 lb. ore contains $75 \times 0.467 = 35$ lb. Fe.

The roasted ore is to contain all of its iron in the form of ferric oxide (Fe_2O_3). It is necessary, then, to determine the per cent of iron contained in Fe_2O_3.

2 atoms Fe = 112 = $\frac{112}{160}$ = 70 per cent
3 atoms O = 48 = $\frac{48}{160}$ = 30 " "
160 100

Thus Fe_2O_3 contains 70 per cent iron. Consequently, the iron content of Fe_2O_3 represents 70 per cent of the weight of the Fe_2O_3.

We found that the 75 lb. pyrite contained 35 lb. Fe. This 35 lb. Fe

at the beginning of this discussion: namely, a slag of 40 per cent SiO_2 and 60 per cent FeO.

The purpose of this illustration is to emphasize the extraordinary effect obtained by submitting an iron sulphide ore to the operation of roasting. Neglecting the copper values for the moment, the original ore contained only pyrite (metallic portion) and quartz (acid earthy portion). This mixture was totally unsuitable for smelting, because the metallic portion would simply melt into a low-grade matte, leaving behind the infusible acid earthy portion. This acid portion could only be rendered reasonably fusible by supplying it with a large proportion of basic earths — according to our present assumption, one and one-half pounds basic earths to each pound of the acid earth.

By means of the roasting process (carried out to an impossible degree of completeness in this illustration), we have converted the metallic portion completely into basic earths, and have supplied the acid earth with even more of such flux than it required.

This is the problem which constantly confronts the copper smelter. His indispensable basic flux (iron) is locked up in combination with sulphur, and is thus unavailable for slag formation. Indeed, it is much worse than useless; for, as it simply melts

then represents 70 per cent of the weight of the Fe_2O_3 which will be formed therefrom. Consequently, there will be $\frac{35}{70} \times 100 = 50$ lb. Fe_2O_3. The iron, however, enters the slag as FeO and not as Fe_2O_3, and must be calculated accordingly. In the 100 lb. of original raw ore — and equally in the 75 lb. of roasted ore — there was 35 lb. of iron. We desire to know what will be the weight of the FeO produced from this 35 lb. of Fe.

1 atom Fe = 56 = $\frac{56}{72}$ = 77.8 per cent
1 atom O = 16 = $\frac{16}{72}$ = 22.2 " "
72 100

Thus FeO contains 77.8 per cent Fe. Consequently, the iron content of FeO represents 77.8 per cent of the weight of the FeO which will be formed therefrom.

The 35 lb. of iron, then, represents 77.8 per cent of the weight of the FeO which will be formed therefrom. Consequently, the FeO will weigh

$$\frac{35}{77.8} \times 100 = 45 \text{ lb.}$$

It happens that the same result can be obtained by a simpler calculation. If we multiply the weight of Fe by the fraction $\frac{9}{7}$, it will give the weight of the FeO produced. Thus, $35 \times \frac{9}{7} = 45$. Or, conversely, if we multiply the weight of the FeO by $\frac{7}{9}$, it will give its content in Fe. Thus, $45 \times \frac{7}{9} = 35$. This is the method commonly used in practice.

down comparatively unchanged (except by the loss of a portion of its sulphur), it dilutes his copper matte and prevents the proper degree of concentration.

By a process of oxidation he burns off the sulphur which renders the iron useless, he converts the iron into an oxide which fluxes his silica, and he produces a small quantity of rich copper matte instead of a great amount of that substance in a low-grade form. The end result is that he has changed the destination of his iron, and sent it into the slag where it is needed instead of into the matte where it is detrimental.

As it is evident that this entire result has been accomplished by oxidation, it follows that it is incumbent upon the student of metallurgy to make himself thoroughly familiar with both the theory and the practice of such processes of oxidation as are applied to sulphide ores of copper.

Oxidation may be effected by various methods and at varying degrees of temperature. It will be natural to begin with the method usually termed "roasting" — the oldest, simplest, and most familiar of this class of processes.

Before entering upon this subject, however, it will be useful to gain some knowledge of the manner in which ores behave at the comparatively high temperatures to which they are subjected in the various metallurgical operations.

CHAPTER IV

THE BEHAVIOR OF ORES AT HIGH TEMPERATURES

THE art of Copper Smelting is based mainly upon the employment of comparatively high temperatures; consequently, the most important technical knowledge which the practitioner of this art can acquire is the knowledge of the behavior of his ores when they are subjected to this unusual degree of heat. Even the accomplished chemist finds much to learn when he enters into a world where temperatures prevail which are from 500 to 1500 degrees C. beyond his accustomed range and where, consequently, a new class of affinities is awakened and a totally unfamiliar set of chemical reactions is produced.

The more important and ever-recurring substances with which the copper smelter has to deal are few in number and comparatively simple in composition; yet, when studied in a fragmentary manner in connection with the various furnace operations, their behavior frequently appears contradictory and perplexing, and leads either to an uncertain perception of the true inwardness of the process involved, or else to a laborious effort to learn by heart just how each mineral may be expected to behave under the peculiar conditions involved in each separate metallurgical operation.

Time and experience may remedy this difficulty and, eventually, bring about the comprehension of certain fixed general laws, applicable to all situations; but this is a slow and costly process, and I believe that it may be shortened and simplified by a systematic, though brief, study of the manner in which each of the ordinary minerals will behave when subjected to any of the conditions which it is likely to encounter during the chain of operations which connects the original ore with the final ingot of marketable copper.

It is manifest that, in a work of this nature, such an exposition must be concise, didactic, and somewhat arbitrary, and that considerations of practical utility must take precedence of extreme scientific accuracy.

The minerals that occur commonly in ores with which the copper smelter is called upon to deal may be classified as follows:

A. Quartz — or silica in any free form (uncombined with a base)

B. Sulphides (subordinately, arsenides and antimonides)

C. Oxides and carbonates

D. Silicates

E. Sulphates

The study of the behavior of these substances, when exposed to furnace temperatures, is complicated by the fact that certain of the metallurgical operations are carried out in an atmosphere containing abundant fresh air (oxidizing atmosphere), while others are conducted in an atmosphere from which practically all of the oxygen has been removed, owing to the presence of other substances which have a great affinity for oxygen, and whose avidity for this gas is still unsatisfied, so that they are likely to take up oxygen from certain of the oxide compounds which may already exist in the ore under treatment (reducing atmosphere). Still a third condition is common, in which the ore undergoes exposure to heat whilst enveloped in an atmosphere from which most of the oxygen has been removed — and which, consequently, is not an oxidizing atmosphere — and which yet contains no actively reducing gases. This is appropriately termed a neutral atmosphere.[1]

The presence, or absence, of oxygen — the most active of all substances at furnace temperatures — modifies profoundly the behavior of most of the more important constituents of the ores which are exposed to these temperatures. This is also true, to a considerable extent, of glowing carbon, or of the carbon monoxide

[1] Some metallurgical writers throw discredit upon this recognition of a neutral atmosphere, pointing out that, even in a reverberatory smelting furnace, the atmosphere varies greatly with the condition of the grate and coal supply, and that at any given moment it may be either oxidizing or reducing according to these conditions. This is quite true; yet in ordinary reverberatory smelting these conditions balance each other pretty nearly, and the final results are usually about such as would occur were the atmosphere consistently neutral throughout the entire operation. It certainly cannot, as a whole, be called oxidizing; nor can we term it reducing. Yet we must provide it with some sort of an atmosphere for descriptive purposes, and, on the whole, I find neutral atmosphere an appropriate term to employ for several of our metallurgical situations.

gas resulting therefrom. Consequently, it is necessary to study our minerals under all three of these conditions; that is to say, when heated in an oxidizing atmosphere, in a reducing atmosphere, and in a neutral atmosphere.

A. — *Quartz, or silica in a free form* (uncombined with any base) SiO.

This mineral, the gangue-rock of most ores, is the most important, because most universal, substance with which the smelter has to deal. It remains passive, unfused, and unchanged in the furnace until a temperature is reached at which its affinities are sufficiently aroused to encourage it to combine with some base (earthy or metallic oxide) with which it is in contact. It then forms, at the point of contact, a silicate — or salt of silica and base — which melts and drips, exposing fresh surfaces of the infusible lump of silica to the action of the dissolving bases. If sufficient base is present, the entire supply of silica gradually enters into combination with the oxide or oxides, and is thus eventually converted into a liquid silicate, or slag. If there is not a sufficient quantity of basic substances (usually oxides of iron, calcium, magnesium, aluminium, etc.) present to satisfy the requirements of the silica as regards the formation of a fusible silicate, a portion of the quartz will be left undissolved, and will remain behind in a solid or pasty condition. The result in practice will be, for the blast-furnace, a general choking, or freezing-up of the charge at the smelting-zone of the furnace-shaft; for the reverberatory, a mass of viscous, half-fused material sticking to the hearth and requiring to be melted free before any proper smelting can be done.

Silica, at smelting temperatures, may be regarded as an infusible anhydrous acid, demanding considerably over one hundred per cent of its own weight of oxides (bases) before it will form a slag suitable for metallurgical purposes. In the majority of instances no such proportion of bases is present in the ore, and it is necessary to supply them from other sources. As silica is the commonest and most abundant mineral in ores of copper, gold, and silver, and as it forms the entire acid side of all copper slags, it follows that two of the most important and universal problems which confront the smelter are:

1. A scientific problem: what amount of bases — as a whole — is required under any given conditions; and what is the least

proportion of each individual base which will suffice to produce a suitable slag?

2. A commercial problem: of all the available bases (either as ores, or as barren fluxes) that may be afforded by the particular district under consideration, which is the most advantageous and economical to procure as the chief basic constituent of the slag to be produced, and how *little* of the more expensive bases need be purchased to comply with the requirements of problem No. 1?

It will be seen, therefore, that quartz, or free silica, is infusible and unchangeable at moderate temperatures, and does not claim the metallurgist's attention in the gentle heat of the roasting furnace or, indeed, until a smelting temperature (say 1000 deg. C.) is attained. From this point it becomes the most active and important constituent of the ore-mixture, combining with any metal-oxide which still exists as an oxide, and forming the sole acid side of the mixture of new silicate-salts which we term slags.[1]

The behavior of quartz, when alone or accompanied by indifferent substances, remains practically the same whether the atmosphere of the furnace be oxidizing, reducing, or neutral; but when accompanied by substances which yield bases on oxidation — such as pyrite — its ultimate behavior is profoundly affected by the condition of the atmosphere in the smelting zone.

B. — *Sulphides* (subordinately, arsenides and antimonides).

Sulphides are prominent constituents of almost every smelting mixture, and their behavior when exposed to heat is simple. It is, however, obscured and rendered peculiarly difficult by the custom of studying each sulphide mineral separately, and attempting to learn its behavior by an isolated effort of the memory.

While it is important that the metallurgist should, in time, become familiar with all the details of behavior of each of his sulphide minerals, it is still more essential that he should, at the very outset, be presented with two or three guiding principles upon which may be based a simple scheme of classification that

[1] Without entering here upon the question as to what rôle is played by alumina in the formation of slags, I believe it to be more convenient to continue to class it as a base in this book.

will enable him to arrange his future details in order, and without burdening his memory with disconnected facts.[1]

The ordinary sulphides which are of interest to the smelter of copper are pyrite, pyrrhotite, chalcopyrite, chalcocite, bornite, covellite, zinc blende, and galena.

The effect of high heat, without oxygen, upon all of these sulphides is to reduce them to their *lowest possible sulphide combination;* that is to say, if — like pyrite — they consist of, or contain, a sulphide possessing more sulphur than their lowest possible sulphide, they lose this excess sulphur, which distils off as vapor of elemental sulphur. One needs, then, only to know what *is* the lowest sulphide form of the metal under consideration, and the result of the heating process is apparent at once.

The lowest sulphide of iron is.....FeS
The lowest sulphide of zinc is.....ZnS
The lowest sulphide of lead is.....PbS
The lowest sulphide of copper is.....Cu_2S

Applying this general rule to the specific minerals under examination, we find that, when exposed to a melting heat in an atmosphere free from oxygen, the following changes take place:

pyrite .....FeS_2 pyrrhotite .....Fe_7S_8	become .....FeS
zinc blende .....ZnS	remains ....ZnS
galena .....PbS	remains ....PbS
chalcocite .....Cu_2S	remains ....Cu_2S
covellite .....CuS	becomes ....Cu_2S
chalcopyrite .....$Cu_2S \cdot Fe_2S_3$ bornite.....$3Cu_2S \cdot Fe_2S_3$	become .....Cu_2S and FeS

We may make even a still more general statement and — representing the metal of the original sulphide by the letter R — may say that, when exposed to high heat without oxygen, the resulting sulphide will have the composition RS — excepting the copper sulphide which will always be expressed by the formula R_2S.

This principle simplifies the study of the furnace operations, as it shows that all the sulphide metals which are likely to be

[1] The general statements that I am going to make in relation to the sulphide minerals refer only to those sulphides which are, ordinarily, of interest to the copper smelter, and may need some occasional slight modification when the detailed consideration of each separate mineral is taken up.

found in copper matte will have the formula RS, excepting the copper itself, which will always correspond to R_2S.

Pyrite, FeS_2, is (technically) the most valuable of all minerals to the copper smelter. Next to quartz, it is the most abundant and universal constituent of his ores, and in a mineralized district it often carries copper, as well as the precious metals; but, entirely apart from the accidental values thus mixed with it, it is usually most welcome to the smelter in at least two directions: first, as a fuel for the blast-furnace, its sulphur and iron yielding a large amount of available heat when oxidized with sufficient rapidity; secondly, as a flux for silica, its iron — when oxidized — forming the smelter's most fusible and most valuable base.[1]

Its behavior is modified profoundly by the condition of the atmosphere in the furnace, and the changes which it undergoes in every possible metallurgical situation are of the greatest importance to the smelter, and form a considerable portion of the foundation of applied metallurgy. This mineral, consequently, demands a more careful and extended study than can be afforded to any other constituent of the smelting mixture.

As the behavior of pyrite depends largely upon the kind of atmosphere which surrounds it when heated, it must be studied accordingly.

Behavior of pyrite (FeS_2) when heated in a neutral, or reducing, atmosphere.—When heated in a neutral, or reducing, atmosphere — such as a retort from which all air is excluded — pyrite undergoes no chemical change until a dull-red heat is attained. It then begins to lose sulphur,[2] and at about 700 deg. C. will have lost $\frac{3}{7}$ of its sulphur, and have become a somewhat indeterminate compound analogous to ordinary pyrrhotite, and which may be considered as Fe_7S_8. This is not an entirely stable compound; for, with increasing heat, it continues to lose a small proportion of its sulphur. It melts at 925 deg. C. without any marked change in its chemical composition excepting the gradual loss

[1] Of course there may be exceptional conditions where too much pyrite is present, and the excess becomes a drawback; but these are exceedingly rare, the normal situation being an excess of quartz and a scarcity of iron.

[2] The sulphur, of course, escapes as a yellowish vapor of sulphur, which can be condensed upon a cool surface. If it meets oxygen while the temperature is still sufficiently high for it to ignite, it will burn to sulphur dioxide gas (SO_2).

of sulphur, and by the time 1200 deg. C. is reached (a temperature at which most slags are liquid), it will have lost another $\frac{1}{3}$ of its sulphur, thus becoming at length FeS.[1]

This is the lowest known iron sulphide, no basic sulphide — such as Fe_2S — being recognized. It follows, therefore, that if the FeS should still continue to lose sulphur, and if there is no oxygen present to oxidize the iron which has thus lost its sulphur, metallic iron must be formed — and this is the case.

This FeS when exposed to increasing heat still continues to undergo slow dissociation, finely-disseminated metallic iron being formed throughout the molten FeS; this being determinable both by the magnet and the microscope. The amount of iron thus formed depends upon the degree of heat attained — which may reach 1500 deg. C. in certain processes — and upon the length of time during which the matte is exposed to high temperatures. All mattes, therefore, which contain FeS may be expected to contain a certain — often very minute — proportion of metallic iron, unless the ores from which they originated were so fusible that the smelting process was unusually rapid, and the temperature of the zone of fusion was low.

The final result, therefore, of heating pyrite in a neutral or reducing atmosphere to high temperatures is ferrous sulphide, often containing a considerable proportion of metallic iron in solution.[2]

Behavior of pyrite when heated in an oxidizing atmosphere. — As the atmosphere in which the pyrite is heated is now to be oxidizing, it follows that the pyrite itself must be in small particles; otherwise, the interior portion of the larger grains would be undergoing the influence of heat alone while excluded from oxygen, as the outer layer of pyrite would act as a protecting envelope.[3]

When pyrite in a pulverized condition is exposed to a dull-red heat and swept by a current of air, so that the spent gases may

[1] This shows the justice of the old-fashioned metallurgical assumption that FeS_2 on smelting in a non-oxidizing atmosphere becomes FeS.

[2] See "Principles of Copper Smelting," Chap. X.

[3] As a matter of fact, if time enough be afforded, the oxygen will eventually penetrate to the centre of even large lumps of pyrite, by means of a series of interesting chemical changes. This is exemplified in the familiar heap-roasting process. This gradual form of oxidation is quite a different matter, and is not contemplated in the present discussion.

be brushed away and ample oxygen may always be in contact with the surface of the mineral particles, it will begin to ignite at about the temperature at which sulphur catches fire, and will oxidize with considerable rapidity, a reddish glow soon spreading over the entire surface, and the temperature rising with such rapidity that care has to be taken to prevent the surface of the charge from melting and hindering the reaction. From what has been stated in the preceding section, it is evident that the volatile sulphur of the pyrite is being driven off by heat alone, just as it would be were no oxygen present; but that, in the presence of air, the dissociated, vaporized sulphur burns at once, with its characteristic blue, flickering flame, to sulphur dioxide gas (SO_2), which escapes from the stack.

The heat thus evolved aids in breaking up the deeper particles of FeS_2 which are not in contact with oxygen, but which furnish their quota of sulphur-vapor to burn to SO_2 when it reaches the surface and finds oxygen with which to combine.

Soon the red-hot FeS, or Fe_7S_8, begins to oxidize, the sulphur burning as always to SO_2, while the iron with which the sulphur was combined is oxidized to such oxides of iron as can exist in a free form under the local conditions; these are the magnetic oxide of iron (Fe_3O_4), and the ferric oxide (Fe_2O_3). If the oxidation were complete, and all of the pyrite present were oxidized to its highest stage of oxidation, the final products would be only two in number: a volatile product escaping from the chimney, which represents all of the sulphur which the pyrite contained originally, and which consists of SO_2 gas, with a small admixture of SO_3 gas; a solid product — Fe_2O_3 — which represents all of the iron of the original pyrite. In practice, this oxidizing operation is never executed in so complete a form, and there are numerous half-way compounds remaining in the mass of oxidized pyrite. These compounds, resulting from the imperfect roasting of pyrite, consist of ferrous sulphates, ferric sulphates, basic iron sulphates, undecomposed FeS, etc., and, from their number and complexity, might be supposed to cause embarrassment to the metallurgist when he is trying later to calculate his smelting mixture and to determine exactly how all the different constituents of his furnace-charge will comport themselves during the melting operation which usually succeeds this process of oxidation. This fear, however, is groundless, owing to the fact that the process of smelting

brings about certain simplifying changes in this mass of heterogeneous compounds. Under suitable furnace conditions, this process of simplification is carried out with complete thoroughness and without attention on the part of the metallurgist, and extends also to the Fe_3O_4 and Fe_2O_3 which form the main bulk of the oxidized pyrite.

These changes are not all completed until the oxidized ore has actually undergone the succeeding process of smelting; but when this has been completed, only two substances will have resulted finally from the long list of compounds which were present in the oxidized pyrite: these are (*a*) ferrous sulphide, FeS, which enters the matte; and (*b*) ferrous oxide, FeO, which combines with the silica of the furnace-mixture and enters the slag.[1]

As his materials pass along through the chain of operations from crude ore to refined copper, the metallurgist is interested in their exact chemical composition (so far as technical or utilitarian purposes are concerned) only at certain fixed points, and does not find it necessary to determine the details of their chemical make-up during the transition developments which occur in passing from one of these fixed points to the next one. For instance: he knows, or should know, the composition of his original ore. His next fixed point will be the matte and slag resulting from the smelting of this ore; and he knows that in order to produce a matte and slag of the exact chemical composition best suited to his purpose he must — at some point between the original ore and the final matte — oxidize such a proportion of the sulphur and iron contents of the ore as science and experience have taught will be necessary for the purpose. He does not care in the least what temporary and evanescent compounds the sulphur and iron may choose to enter into during this oxidation, nor what changes they may undergo during the earlier, preparatory stages of the smelting operation. What he *does* want to know is how

[1] It is of the greatest importance to recognize that this ferrous oxide which enters the slag cannot exist by itself, and hence was not present (as such) in the oxidized pyrite. It results solely from the reactions which occur in the smelting process, and is produced (so far as it is advisable to assume at present) from the ferric and magnetic oxides which form the bulk of the oxidized pyrite. Under suitable conditions, at a smelting temperature, and in contact with heated silica, the higher iron oxides — Fe_3O_4 and Fe_2O_3 — are changed into ferrous oxide — FeO — which, at the instant of its formation, unites with the silica to form a ferrous silicate (slag).

much of his iron will remain combined with sulphur and go into the matte, and how much of it will form FeO and go into the slag.

This custom of sweeping away complications, and of reducing (in one's calculations) all of the iron compounds to (*a*) what will eventually become FeS for the matte, and (*b*) what will eventually become FeO for the slag, is eminently useful, and sufficiently accurate for all ordinary purposes. Indeed, in rough calculations, this process of simplification may be carried even further, and pyrite may be figured as consisting of one-half iron and one-half sulphur. As one-half of this sulphur, approximately, is lost by heating (without oxidation), the resulting iron sulphide may be assumed to have three-quarters of the weight of the original sulphide. In practice, a correction must be applied to this result, as will be explained later.

It may, then, be said that the complete oxidation of pyrite yields only two products: the SO_2 gas which escapes from the chimney and is thus removed from the process; and Fe_2O_3; but that this perfect result is never attained nor desired in ordinary metallurgical oxidation, and that a variety of temporary between-compounds is formed, which may in practice be disregarded by assuming that the end-products of the preparatory oxidizing process — after they have been modified and sorted by the action of the smelting furnace — may be looked upon as only two in number, namely, FeS, for the matte; FeO, for the slag.

Pyrrhotite, Fe_7S_8 — Heated without oxygen. It will be noted that this mineral has the same chemical composition that pyrite has after the latter has been heated to 700 deg. C. and has lost three-sevenths of its sulphur. (See p. 44.) Therefore we need not pursue further the examination of pyrrhotite when heated in a non-oxidizing atmosphere, as it must, of course, be identical with the behavior of the analogous compound resulting from pyrite.

This same analogy also exists when pyrrhotite is heated in an oxidizing atmosphere. It behaves simply as pyrite does after the latter has lost nearly one-half of its sulphur; the final result of complete oxidation being SO_2 gas and Fe_2O_3.

Chalcocite, Cu_2S. — This mineral, being already at its lowest possible combination with sulphur, undergoes no chemical change when heated in a neutral or reducing atmosphere.

In an oxidizing atmosphere, and under gradually increased heat and ideally favorable conditions, it forms successively cupric

sulphate, cuprous oxide, and cupric oxide. This being its highest oxide, it undergoes no further change. Its sulphur content escapes mainly as SO_2 gas.

Covellite, CuS. Exposed to heat without oxygen, it loses half of its sulphur and becomes Cu_2S. (See Chalcocite.)

Exposed to heat and air, it passes through practically the same changes as chalcocite, becoming eventually cupric oxide.

Chalcopyrite, $Cu_2S{\cdot}Fe_2S_3$. In studying the behavior of these compound sulphides of copper and iron, it is convenient to dissect them into two parts: the copper sulphide and the iron sulphide. The copper-sulphide molecule, being already the lowest combination of copper and sulphur, can lose nothing more when exposed to heat without air, and behaves simply like chalcocite. The iron-sulphide molecule, having more sulphur than corresponds to the lowest combination of iron and sulphur, will — at a high temperature — lose sufficient of its sulphur to become FeS: that is to say, it will lose one-third of its sulphur and become Fe_2S_2 — or FeS.

Consequently, according to theory, chalcopyrite when exposed to high heat without air will lose one-third of the sulphur belonging to the Fe_2S_3; this is 8.72 per cent of the original weight of the chalcopyrite; or, as is usually assumed by the metallurgist in rough calculations, the chalcopyrite will lose 10 per cent of its weight when exposed to a melting heat in a neutral or reducing atmosphere.[1]

[1] This calculation is given in full as an aid to those who are not skilled in metallurgical arithmetic.

Problem. What loss of weight will chalcopyrite experience when subjected to heat without the presence of oxygen?

Data to be used. Chemical formula of chalcopyrite, $Cu_2S{\cdot}Fe_2S_3$. Assume that the only loss experienced will be one of the atoms of sulphur belonging to the iron. Atomic weights: Cu, 63.6; Fe, 56; S, 32.

Solution. Find first the percentage of copper, sulphur, and iron in chalcopyrite.

2 atoms	Cu @ 63.6	=	127.2
4 atoms	S @ 32	=	128
2 atoms	Fe @ 56	=	112
	Total		367.2

the proportion of copper will be	$\frac{127.2}{367.2}$	=	34.64 per cent
the proportion of sulphur will be	$\frac{128}{367.2}$	=	34.86 " "
the proportion of iron will be	$\frac{112}{367.2}$	=	30.5 " "
			100. per cent

Bornite, $3Cu_2S \cdot Fe_2S_3$. When exposed to heat, either with or without oxygen, we may assume that bornite yields the same end-products as chalcopyrite.

Arsenides and antimonides. These minerals, when they occur at all with copper ores, are usually present in such subordinate proportion, that their reactions are profoundly modified and obscured by the great quantity of pyrite and other sulphides which form the bulk of the ore.

It will suffice for present purposes to say that, whether heated in an oxidizing or in a reducing atmosphere, it is practically impossible to remove all of the arsenic and antimony. A considerable portion of those elements still remains in the ore, either combined direct with the metal, or as basic metallic arsenides and antimonides.[1]

C. *Oxides and carbonates.* When heated in an atmosphere free from oxygen, the ordinary metallic oxides which interest the copper smelter undergo no important chemical change, as none of them are dissociated from their oxygen by heat alone, except at temperatures far above those under consideration.

When heated in an actively reducing atmosphere, the behavior of one or two of the oxides becomes more interesting. At a

In 100 lb. of chalcopyrite
the copper will weigh ..... 34.64 lb.
the sulphur for the copper 8.71
the cuprous sulphide weighs ..... 43.35 lb.
the iron will weigh ..... 30.5 lb.
the sulphur for the iron 26.15
the iron sulphide weighs 56.65
100.

The cuprous sulphide does not change. The Fe_2S_3 loses one-third of its sulphur, becoming Fe_2S_2 (FeS).

The Fe_2S_3 contained 26.15 lb. of sulphur. It loses $\frac{26.15}{3} = 8.72$ lb. sulphur, or 8.72 per cent of the total weight of the chalcopyrite.

In rough calculations this loss is usually figured as 10 per cent, to which must be added the additional loss due to a certain amount of oxidation which always takes place in actual practice, and which varies according to local furnace conditions.

[1] An exception to this general rule must be noted in the case of several of the more volatile of the substances which we have just been studying. In the violent bessemer oxidation of pyrite smelting, or of converting, zinc, lead, arsenic, and antimony may — under favorable conditions — be almost completely volatilized and removed.

moderately elevated temperature, the oxides of copper and lead are reduced to metal, the former combining eagerly with sulphur, to form Cu_2S. Lead, however, is inclined both to volatilize and to slag.

The behavior of the iron oxides is of still greater importance, as the entire technical success of the smelting process depends upon these oxides. They are usually present in the furnace-charge as ferric and magnetic oxides. These are relied upon to furnish the ferrous oxide (FeO) which is the mainstay of the slag to be formed, and which is the only suitable oxide of iron for the purpose, as the higher iron oxides form unsatisfactory compounds with silica.

If these higher iron oxides should be exposed to long and powerful reducing action at a high heat, they would lose *all* of their oxygen and become metallic iron — as in the smelting of iron ore.

The copper smelter, however, desires to limit the reducing action of his process to just the point where the higher oxides will have become reduced to FeO, and then to provide this ferrous oxide with the proper quantity of white-hot silica. Under these conditions, the FeO will combine with the silica to form the new silicate (or mixture of silicates) which is termed *slag*.

The oxides of manganese follow so closely the behavior of the oxides of iron that they are commonly figured as part of the latter.

Zinc oxide goes in all three directions: that is to say, a portion is volatilized, a portion combines with sulphur and enters the matte, and a portion remains in its oxidized form and goes into the slag. It causes trouble and loss in each of these directions, and is always objectionable.

The oxides of the earths undergo practically no change until a high temperature is reached. Then, in the presence of silica, they combine with that acid, forming slag.[1]

All of the carbonates that are of interest in the present connection lose their carbon dioxide at a moderate temperature, and become oxides. (See preceding paragraph.)

D. *Silicates.* — The only important silicate that presents any marked peculiarities when submitted to heat is the mineral

[1] A certain proportion of sulphates or sulphides — according to the atmosphere of the furnace — is formed from most of the earthy oxides by reacting with the sulphides present. This is only exceptionally of practical importance.

chrysocolla, or cupric silicate ($CuSiO_3 + 2H_2O$). In an oxidizing atmosphere and when kept apart from reducing agents — such as sulphur or carbon — it appears to undergo little change (excepting the loss of its water) until a temperature is reached at which the affinity of silica for bases is awakened. At this point, and in the presence of the ordinary bases, chrysocolla is decomposed, its silica uniting with the stronger bases, and the residual copper oxide being reduced by carbon or sulphur to metallic copper or cuprous sulphide.

The silicates with which the copper smelter usually has to deal are the ordinary complex silicates of alumina, iron, lime, magnesia, etc., which are the components of the rock-forming minerals. It would not be profitable, even were it feasible, to study minutely the series of changes which these substances undergo when exposed to metallurgical temperatures. A few general principles will serve to indicate their behavior so far as practical results are concerned.

It may be said, in a general way, that: — (*a*) they are not decomposed by heat alone, nor do they suffer any important chemical change until a high temperature is attained; (*b*) when a melting temperature is approached — perhaps between 950 deg. and 1300 deg. C., according to the nature of the particular silicate under observation — they begin to soften and to manifest an affinity for such substances as are lacking in their composition to place them in a state of chemical equilibrium (or neutrality) corresponding to the conditions which they are experiencing at the moment referred to. There are various influencing and modifying conditions that affect the precise chemical combination which the laws of nature are striving to accomplish. The most important of these is the degree of temperature to which they are exposed.

Most of the natural silicates contain too much silica to melt at the temperature of the ordinary copper furnace, and are anxious to take up sufficient additional bases to neutralize the excess silica and to reduce the silicate-degree of the compound to a lower and more fusible stage. As these refractory silicates are not fully liquid at the prevailing temperature of the furnace, they cannot flow about in search of the desired bases; yet propinquity, to the extent of absolute contact, is essential before the desired reactions can take place. As they cannot seek the bases,

the bases must be brought to them, and this is what actually takes place in the smelting of all ores which contain minerals that are infusible at our ordinary smelting temperatures.

In the reverberatory furnace this actual contact is effected by having the constituents of the charge (especially the infusible portions) in a more or less pulverized and mixed condition. This condition of propinquity is still further aided by the boiling and bubbling of the charge during the period of fusion, and by the free floating of the same upon the pool of molten matte with which the hearth should be filled, and which, in recent times, has taken the place of the tedious and expensive rabbling of the half-melted charge which was an essential part of the older practice when smelting difficult ores.

In the blast-furnace, where comparatively large lumps of infusible material often form a considerable proportion of the mixture, the necessary propinquity is attained mainly by a process of liquation. The more refractory lumps sink — as the whole column of ore sinks — until they reach the bed of glowing coke, at the tuyeres, which rests upon the liquid contents of the crucible. They are already softened, and are diminishing constantly in size as their outside layers are gradually fluxed away. They cannot, however, sink further, for, as soon as the coke below them is consumed, they would block and freeze the furnace unless one of two conditions prevailed: either the more fusible materials showering down upon them (liquating) from the column above must furnish the bases necessary to combine with the excess silica, and thus flux it into the condition of a well-melted slag, or the temperature produced by immediate contact with the layer of glowing coke must be high enough to melt them into a condition of sufficient liquidity to enable them to flow out of the furnace, and thus permit the uninterrupted descent of the superincumbent ore-column. [1]

It may, then, be assumed that the ordinary silicates pass practically unchanged through the moderate heat of the roasting process, and undergo but little modification in the smelting furnace itself, until a temperature is approached at which they

[1] The above statement, while portraying the practical results which accompany the smelting of ordinary ores, must not be received as a complete explanation of the process of slag-formation. This is an exceedingly complicated operation, and still requires much research.

begin to soften and to become ready to combine with other substances which may be placed in contact with them, providing these new substances are of a nature to correspond to the needs of the softened silicates in forming more fusible compounds. It need scarcely be added that these new substances are almost invariably of a basic nature, and that they commonly consist for the most part of ferrous oxide, lime, magnesia, and alumina.[1]

E. *Sulphates.* With the exception of heavy spar (barite, $BaSO_4$), the occurrence of any considerable proportion of sulphates in the original ores received at a copper smelter is uncommon. Owing, however, to the reactions which take place during the roasting process (and sometimes in the upper zones of the blast-furnace itself), a considerable quantity of sulphates may be present in the ore-mixture by the time the process of actual fusion begins.

Iron sulphate (ferrous sulphate, $FeSO_4$) is formed freely when an iron sulphide is exposed to a dull-red heat with free admission of air. This is an unstable compound at more elevated temperatures, and is so easily decomposed by heat alone that it requires little attention. When broken up, its sulphur content passes off as sulphur dioxide gas (with a little sulphur trioxide), while its iron becomes eventually Fe_2O_3. In a strongly reducing atmosphere — as in the coke-burning blast-furnace — a portion of its sulphur passes off as SO_2, its FeO combining with silica to form slag, while another portion is reduced to FeS which enters the matte.

In a more nearly neutral atmosphere — as in reverberatory smelting — beneficent reactions occur between the $FeSO_4$ and unroasted particles of FeS, by which sulphur is eliminated as SO_2: thus, $3Fe{\cdot}SO_4 + FeS = 4FeO + 4SO_2$, it being assumed that silica is present to combine with the FeO as it is formed.

Cupric sulphate ($CuSO_4$) is also formed during the roasting of sulphide ores, and is also decomposed by heat in the same manner as ferrous sulphate, although requiring a somewhat higher temperature.

In the smelting of a mixture containing unoxidized sulphides, it acts in the same advantageous manner as ferrous sulphate, thus: $4CuSO_4 + Cu_2S = 6CuO + 5SO_2$. The copper oxide re-

[1] For reasons elsewhere explained I believe it to be more convenient to class alumina among the bases.

sulting from this reaction combines with more sulphur to form Cu_2S and enters the matte.

The alkaline earths also form sulphates, to some extent, during the operation of roasting, and thus become common constituents of the smelting mixture.

Calcium sulphate, $CaSO_4$, is formed freely during the roasting of sulphide ores containing calcium carbonate. Exposed to higher heat in an oxidizing atmosphere, it remains unchanged until 1200 deg. C. is reached, when dissociation begins, and increases as the temperature rises. It is not, however, until actual fusion takes place — at about 1360 deg. C. — that the dissociation is completed, the products being CaO, SO_2, and O.

In the presence of free silica, the decomposition of calcium sulphate is completely effected at a somewhat lower temperature than the above. The silica displaces the sulphur trioxide and appears to form a calcium bisilicate at a temperature of about 1250 deg. C., which is 262 deg. lower than the melting-point of the same compound.[1]

In a reducing atmosphere, a certain proportion of the calcium sulphate is changed to calcium sulphide (CaS), which is dissolved to some extent in both slag and matte. It is not often, in practice, that any serious inconvenience is experienced from this substance.

Magnesium sulphate ($MgSO_4$) is a less stable salt than calcium sulphate and, even if formed freely during the earlier stages of roasting, is decomposed to a considerable extent during the closing period of the same operation. In a reducing atmosphere, and at a melting temperature, magnesium sulphate is mostly decomposed, yielding MgO, which enters the slag.

Barium sulphate ($BaSO_4$), when present, usually appears as the mineral barite — or heavy spar. This is a peculiarly stable sulphate and undergoes no change in the oxidizing atmosphere of the roasting furnace nor, indeed, until a temperature of 1580 deg. C. is attained, when it melts and undergoes partial dissociation.[2]

In the presence of silica the decomposition of the barite and

[1] Hofman and Mostowitsch, "The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes," in *Trans. Am. Inst. Mining Engineers*, xxxix, 628.

[2] Doeltz-Mostowitsch, *Zeitschrift für anorganische Chemie*, May 28, 1907.

the fluxing of the resulting BaO take place at a temperature some 200 deg. C. lower than above, although the resulting silicate is not fluid at that temperature unless other bases are present.

When melted in an oxidizing atmosphere with both silica and iron sulphide, the decomposition of the barium sulphate may be complete, the BaO forming a silicate, while the SO_3 escapes as SO_2 and O. These are the reactions which make the smelting of heavy-spar ores so much more satisfactory in the reverberatory furnace or the pyrite furnace than in the coke-burning blast-furnace. In the latter, much of the barium sulphate is reduced to barium sulphide (BaS), which, owing to its high specific gravity and its want of liquidity, is apt to form a semi-fused product, half way between slag and matte, that occasions great trouble and loss.

CHAPTER V

ROASTING

It was pointed out in a preceding chapter that in a heated atmosphere, free from available oxygen, the sulphides of most metals melt into a matte with but little loss of weight. This behavior is in direct opposition to the ordinary purpose of the smelting process, which is to collect the valuable metals in a *small* amount of matte.[1]

It was further stated that silica combines only with the *oxides* of metals to form slag, and that, therefore, iron — our most valuable basic flux — is useless for slag-forming purposes while it remains in combination with sulphur.

These two fundamental propositions led to the obvious conclusion that, in order to diminish the quantity of matte that would form during the operation of smelting, it was necessary to drive off a large proportion of the sulphur which was united with the metals as sulphides; and that, in order to obtain the iron in a condition suitable for combining with silica, it was essential to convert it into an oxide. These two important and distinct results are attained by a single process termed *roasting*.[2]

A brief study of the principles of roasting is essential to the intelligent comprehension of its practice.[3]

[1] In both blast-furnaces and reverberatory furnaces, as ordinarily operated, the carbonaceous fuel consumes most of the oxygen of the air which is present, leaving little oxygen available for any combustion of sulphides. This is the condition which I am assuming in the present discussion. The practice of introducing an excess of air for the purpose of oxidizing the sulphides during the smelting itself is quite a different matter, and will be considered later.

[2] The term *roasting*, as used in America, must not be confounded with a different important operation bearing the same name in English metallurgical literature. English writers apply the term *roasting* to a stage of matte-refining where pigs of high-grade matte are subjected to a slow oxidizing melting for the production of a crude metallic copper called blister copper. I shall designate this latter process as *blister roasting*.

[3] See "Principles of Copper Smelting," chapters III and V.

It will be useful to consider two primary conditions essential to the oxidation of ordinary sulphides with maximum rapidity:

(*a*) Oxidation demands a temperature sufficiently elevated to excite affinity between the oxygen and the sulphide particle to be oxidized, without being so high as to melt the surface of the fusible sulphide, and thus form a protecting layer upon it, which diminishes further reactions to a most striking degree.

(*b*) Oxidation demands a constant and abundant supply of oxygen in immediate contact with the surface which is to be oxidized.

As these two statements cover pretty much the entire requirements for rapid and economical roasting, it is worth while to examine them in detail.

(*a*) Oxidation demands a temperature sufficiently elevated to maintain the desired reactions, without being so high as to melt the surface of the sulphide particles. Each one of the metallic sulphides has its own individual melting-point and its own degree of temperature at which the reactions between the oxygen and the sulphides take place with the greatest rapidity and in the most thorough and effectual manner.

The ordinary sulphides with which the copper smelter has to deal are pyrite (FeS_2), pyrrhotite (Fe_7S_8), chalcopyrite ($Cu_2S \cdot Fe_2S_3$), bornite (Cu_3FeS_3), chalcocite (Cu_2S), blende (ZnS), and small amounts of galena (PbS).

Chalcocite and bornite may be eliminated at once as being so rich in copper that they need little consideration in the roasting process; and galena may also be disregarded, as containing only 13 per cent sulphur, and being present in too small proportion to affect the operation materially. This leaves only pyrite, pyrrhotite, chalcopyrite, and zinc blende to consider as the essential sulphides of ordinary ores of copper; while any individual ore is likely to contain only two or three of these four minerals.

These four minerals, then, must be the chief object of our attention in roasting copper ores. It will be remembered that we are, as yet, speaking solely of the temperature to which the ore should be subjected in order to obtain its most rapid and efficient oxidation.

Before we make an attempt to fix the exact temperature which will be most suitable for roasting, we may — in practice — make a still further elimination of the minerals which are to be con-

sidered. For instance: zinc blende seldom occurs in copper ores in any very considerable proportion; and, even when it does, its melting-point is so high that there is not the slightest danger of its causing trouble at the beginning of the roasting process. It may, however, resist thorough roasting, and may sometimes cause annoyance at the end of the roasting, from the pertinacity with which it holds on to its sulphur; but, at present, we are speaking only of the initial stage of roasting, and may disregard the blende completely.

Even chalcopyrite seldom occurs in quantity sufficient to exert any profound influence upon the operation of roasting. The main influence that it would exert, were it present in very large proportion, would be to lower the melting-point of the charge, and thus demand a lower initial temperature.

This leaves only pyrite and pyrrhotite, and as the latter is comparatively rare, whilst the former constitutes the almost universal sulphide mineral with which copper smelters have to deal, we may simplify the study of proposition (*a*) by concentrating our attention for the present upon pyrite.

This proposition demands a temperature sufficiently elevated to ignite the pyrite rapidly, and yet not so high as to melt the surface of the sulphide particles. This temperature may be maintained in one of two ways:

1. By the burning of carbonaceous fuel, and the consequent production and maintenance of the desired temperature mainly from this source — though always with considerable aid from the heat resulting from the oxidation of the sulphide itself.

2. By the heat derived from the oxidation of the constituents of the ore itself, without the aid of carbonaceous fuel, heated air, or any extraneous source.

Before it is possible to decide which of these two methods to adopt, the local conditions must be considered with great care. The chief points which determine whether or no an ore may be advantageously roasted by the heat derived from its own oxidation are:

Whether the ore contains a sufficient proportion of free-burning sulphides to maintain the vigorous combustion essential for its own roasting.

Whether the intention of the process is to remove nearly all of the sulphur in the ore, or merely to remove a considerable

proportion of it, leaving several per cent of residual sulphur in the roast.

Provided that there is a sufficient amount of free-burning sulphides (usually pyrite) in the ore to maintain vigorous combustion until a considerable proportion of the sulphur has been removed; and assuming also that the roasted ore is to be submitted to the usual matte-smelting operation, and thus requires only a partial roast, we have then the conditions suitable for roasting without the aid of outside heat.

The degree of temperature required for the process, and the apparatus whereby the somewhat scanty supply of heat furnished by the oxidation of the sulphides is conserved and applied, is so involved with the technical details of the operation itself, that it may be studied to better advantage in connection with the next proposition.

(*b*) Oxidation demands a constant and abundant supply of oxygen in immediate contact with the surface which is to be oxidized.

This apparently simple demand covers a number of important points, and lies at the foundation of our modern mechanical roasting furnaces.

If ordinary pyrite is crushed through a screen with, say, 200 meshes to the inch, and this material is freed from the still finer dust by passing it over a 250-mesh screen, we shall have a product consisting of minute grains ranging in size from 200 to 250 mesh. Submitting this material to a thoroughly oxidizing atmosphere, at a suitable temperature, the removal of the sulphur — practically speaking — is instantaneous and complete. This may easily be demonstrated by showering a thin stream of the fine pyrite into a heated shaft under suitable precautions. In this highly oxidizing atmosphere the sulphur escapes from the stack mainly as sulphur dioxide, with a little trioxide, while the iron falls on the heap at the bottom as a mixture of magnetic oxide and ferric oxide. Naturally, this rapid oxidation generates a large amount of heat.[1]

If a similar trial is made, under identical conditions, with

[1] Many experiments of this nature were conducted by the late Horace F. Brown in connection with his "down-draft roasting furnace" which he was working on at the time of his death. At his request, I checked much of his work, and was struck with the thoroughness of the roasting when the particles were sufficiently fine.

pyrite grains of a size between 150 and 200 mesh, the amount of residual sulphur in the roast, though still very small, becomes nevertheless quite appreciable. A microscopic examination of the product shows that this increase in sulphur is due mainly to an imperfect roasting of the larger grains of pyrite, a certain proportion of them showing kernels of unoxidized sulphide, although modified to a composition apparently approaching pyrrhotite (Fe_7S_8).

A similar trial with 50 to 100-mesh stuff will disclose many particles which have undergone only this surface-roasting, and enclose a comparatively large kernel of this modified sulphide. The per cent of sulphur in the roasted product is now considerable, and a further increase in the size of the grains is followed by a disproportionate increase in the residual sulphur.

The commercial aspect of the case — that is to say, the most suitable size to which an ore should be reduced to effect the most profitable roasting — will be studied in connection with the process itself.

Continuing with the examination of proposition (*b*), we may, by the mere process of reasoning, infer that, for the most rapid and efficient roasting, each particle of sulphide must be constantly in actual contact with heated air; moreover, this air must be reasonably pure, and not too greatly mixed with foreign, non-oxidizing gases, such as sulphur dioxide and trioxide, products of the combustion of coal, and the like. Experiments as well as experience emphasize the depressing effects which follow the dilution of the air with any such products.

It is probable that the oxidizing power of air loses its effect out of all proportion to the extent of its dilution. In plainer language, if the air in our roasting furnace is diluted by foreign gases until it contains only 75 per cent of its normal oxygen, it will not exert 75 per cent of the oxidizing effect of pure air, but will fall far below this point. This shows forcibly that a sufficient draft to remove the non-oxidizing gases, and to bring fresh oxygen, is of primary importance in the operation of roasting.

There is still one more point to consider in connection with the question of bringing heated oxygen into contact with each sulphide particle. As will presently be seen, experience has taught us that the most suitable and economical method for effecting this purpose is to pass a current of heated air over large, com-

paratively thin, layers of more or less fine ore. Under these conditions the *small* surface-particles oxidize with great rapidity, the larger grains reacting slowly, as we have already noted. If we were to wait for the oxidizing reactions to make their slow progress to the very center of these larger sulphide grains, the process of roasting would be prolonged to an impracticable extent, and commercial considerations would require a complete change of method.

The copper smelter, however, does not require a complete roast, as a certain amount of sulphur in the smelting charge is essential to the production of matte. Consequently, as soon as a reasonable proportion of the finer grains have had their exposed surfaces so far oxidized as to lessen materially their affinity for oxygen, he finds it expedient to replace them by fresh sulphide particles brought up from the deeper unoxidized layer. This stirring of the ore is effected at brief intervals, and is continued until it has lost sufficient of its sulphur to be in suitable condition for the succeeding smelting process.

In practice, the material to be roasted often consists of pyritous concentrates, screenings from first-class ore, and other moderately fine sulphides, the individual grains often ranging from one-half inch (12 mm.) in size — and even larger — down to impalpable powder. As delivered at the roasting furnace it may contain, perhaps, 25 to 40 per cent sulphur, and it is usually required to roast it to such an extent that the product shall not contain more than seven or eight per cent of this substance.[1]

As the larger fragments of sulphides will undergo but a very imperfect roasting during their passage through the furnace, it is plain that the finer particles must be oxidized to a point where they shall contain considerably less sulphur than the seven or eight per cent which is to remain at the termination of the process. In

[1] It must be borne in mind that as an ore usually loses weight in roasting, the percentage of sulphur found in the roasted product cannot be compared directly with the percentage of sulphur that was contained by the original ore. For instance: a certain sulphide ore contained 34.2 per cent sulphur. Eighty pounds of this ore, roasted in a small furnace, yielded a product weighing 66.4 lb., and containing 8.8 per cent sulphur. What percentage of the total sulphur was removed by the roasting?

Total sulphur in original ore ....80 × 0.342 = 27.3 lb.
Total sulphur in roasted ore....66.4 × 0.088 = 5.8 lb.
Sulphur eliminated = 21.5 lb.
$\frac{21.5}{27.3}$ = 78.7 per cent sulphur removed by roasting.

other words, we must over-roast the fine particles in order to offset the undue amount of sulphur which remains in the coarser lumps. This plan answers perfectly well in practice, and is decidedly more economical than it would be to attempt to separate the raw ore, by screening, into a coarse and a fine portion; or to crush the coarser portion down to a size suitable for efficient roasting, and then subject it to a merely partial oxidation.[1]

Sulphide ores vary very materially as to their *permeability* to the oxidizing process. Exposing two lumps of pyrite, from different districts, to identical oxidizing influences for six hours, and then making sections in such a manner that the thickness of the partially oxidized crusts could be observed, I found that in the more compact specimen, the surface layer of material that had plainly been affected by the treatment was somewhat less than half as thick as in the more porous specimen.

While such experiments are interesting and instructive, they are not essential to the practical management of the roasting plant, which may be guided by the following simple rule: when a mixture of coarse and fine sulphides takes so long to roast down to a desired per cent of sulphur that the furnace is manifestly making a smaller output per square foot of hearth-area than it is accustomed to do on ores of the same general chemical composition, rough tests should be made to determine whether the coarser sulphide particles are retaining too much sulphur. If this is the

[1] The contrasting sulphur content of the fine and the coarse particles, after roasting, is shown by a rough test made by myself at the Parrot smelter in Butte, in 1884. The roasting was conducted in long hand-reverberatories, the raw charge containing 36.6 per cent sulphur, and consisting of mixed pyritous concentrates, ranging from three-quarter inch (19 mm.) in diameter down to the finest material produced from fine jigs and vanners. In the following test, a 24-hour sample of the roasted ore contained 8.8 per cent sulphur. A given weight of this sample was passed over a screen having 6 meshes to the linear inch (3 mm. openings), care being taken to pulverize between the fingers all porous lumps which were manifestly only agglomerated particles of fine ore. Fifty-four per cent of this roasted material passed through the openings, 46 per cent remaining upon the screen. The fine material contained 4.4 per cent sulphur; the coarse material contained 13.7 per cent sulphur. The results check as closely as can be expected:

100 pounds roasted ore containing	8.8	lb. sulphur
54 lb. fine ore at 4.4 per cent sulphur =	2.38	lb. sulphur
46 lb. coarse ore at 13.7 per cent sulphur =	6.3	lb. sulphur
100 lb.	8.68	lb. sulphur

case — as will be found in the great majority of instances — a sufficient proportion of the coarse sulphide particles should be held out from the roasting charge. The disposition that should be made of these coarser particles is a matter of unusual importance, and will be considered fully in a later chapter.

The foregoing arguments are sufficient to emphasize the fundamental importance of removing the partially roasted layer which blankets the deeper sulphides, and of bringing these latter to the surface where they, in their turn, may have a chance to exercise their affinity for the oxygen of the air-current which is passing over the hearth of the furnace.

This stirring of the ore during roasting was formerly effected by hand labor, employing long paddles or rabbles introduced through the side doors. It is a laborious, tedious, and imperfect operation, demanding more than ordinary strength and faithfulness, and requiring a large number of men in proportion to the output. It may now be considered, in the main, as belonging to the history of the past, and those who may be forced, by exceptional surroundings, to use it for a time, will find ample details of practice in the older metallurgical treatises. I will merely state that at Butte, Montana, the cost of roasting pyritous concentrates in long hand-reverberatories from 40 per cent sulphur down to 8 per cent was $1.50 to $1.80 per ton of original sulphides ($1.65 to $1.99 per metric ton). These reverberatories had a hearth-area of about 1000 square feet (93 square meters), and could roast 13 tons (11.7 metric tons) of raw sulphides per 24 hours, consuming 2.6 tons (2.25 metric tons) of inferior coal. The output per 24 hours per square foot of hearth surface was, therefore, 26 lb. (127 kg. per square meter), and the ratio of ore to fuel was 5.2 to one.

The weakest point of these hand roasters was their lack of means for the frequent stirring of the ore. Experience showed that it was not economical to employ more than two — exceptionally, three — men on these long furnaces. Even with eight-hour shifts, and under the stimulus of contract-roasting, it was impossible for the men to stir thoroughly more than one hearth each (of the four-hearth furnaces) every 30 minutes; indeed, only exceptional men under exceptional conditions could maintain this standard. This means that the entire contents of the furnace would receive a thorough stirring once every hour; and as each individual square foot of any hearth was not receiving atten-

tion for more than one-third of the time during which that hearth was being stirred, it is clear that the individual sulphide particle could not expect to be moved for more than 10 minutes in each hour. Thus, during 50 minutes out of each hour the ore remained unmoved, and undergoing but little influence from the roasting process.

For experimental purposes, I have doubled — and even trebled — the number of workmen at such a furnace; and, while the output was increased largely, the cost per ton of the operation increased in a still greater ratio. This resulted from disturbing factors foreign to the oxidizing process itself; among these were: undue cooling of the furnace from too many open doors in the hearth; interference of the long tools; great decrease in the average efficiency per man, owing to the impossibility of obtaining so large a number of efficient workmen; and various other causes.

As the mines increased their output and made greater demands upon the smelters, it became evident that an apparatus which was so designed that it acted efficiently during only about one-sixth of the time, and lay practically inert during the other five-sixths, was not in harmony with the spirit of the age. The time was ripe for any reasonable change.

The gradual substitution of the revolving cylinder for the hand roaster effected a marked economy, especially in the United States, where labor conditions are such that the employment of mechanical methods is peculiarly advantageous. This type of furnace, however, is not well suited to the high temperature, rapid driving, and cheap handling demanded by the copper smelter, whose (designedly) imperfect roasting of large quantities of ore high in sulphur and consisting mainly of sulphide grains of mixed sizes — from that of a walnut down to slimes — is best carried out with a thin layer of ore, traversed by comparatively slow-moving plows.

It was learned that, on the whole, the time-honored reverberatory roaster offered the most advantageous conditions, provided only that the oxidized blanket of particles could be removed from the surface with sufficient frequency, while fresh sulphides were brought up from below to replace them. In other words, the situation was resolved pretty much into a single problem: to retain the general features of the old reverberatory hearth, but to add to this a very much more frequent and thorough stirring of the ore.

This latter demand, as already demonstrated, can be met only by the substitution of mechanical devices for hand labor.

In 1850 an Englishman, named Parkes, invented and operated a mechanical roasting furnace which possesses peculiar interest for the copper smelter, as it embodied pretty much all of the principles — and many of the more important details — of the successful automatic roasters of to-day.

In the accompanying illustration from Plattner's "Allgemeine Hüttenkunde" of 1860 may be seen the superimposed hearths;

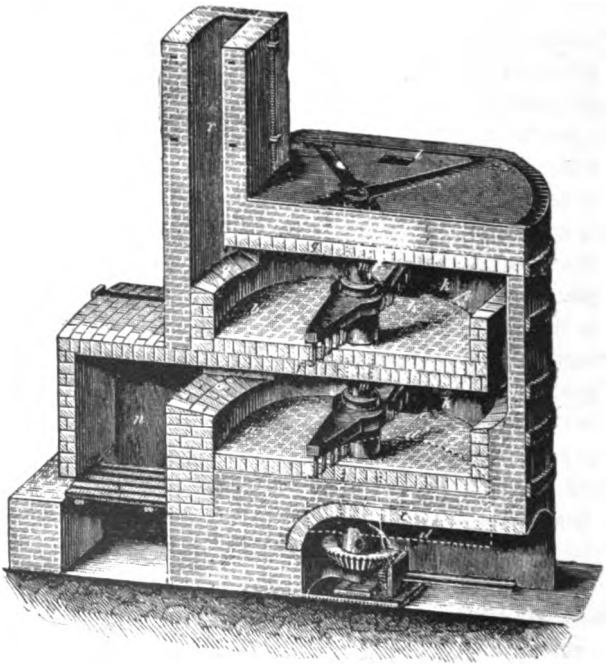


FIG. 8. — Parke's roasting furnace

the vertical, hollow, air-cooled, central shaft, with its bevel gear; and various other important details familiar to modern engineers. The hearths had a diameter of 12 feet, and the capacity of the furnace was about 11,000 lb. per 24 hours of heavy pyrite fines, roasted down to three per cent sulphur. The work of the furnace was highly satisfactory, but the rapid destruction of the brickwork caused the abandonment of an apparatus in which lay the germs of our most successful modern roasting furnaces.

In 1873 McDougall brought out, in England, a modification of Parkes' mechanical roaster, consisting of six superimposed circular hearths enclosed by cast-iron cylinders. The furnace resembled a boiler standing on end, six feet in diameter. It contained a solid, vertical cast-iron shaft to which were attached the six sets of arms for carrying the stirrers, and the stirring operation moved the ore radially across each hearth, until it dropped upon the floor next below and was eventually discharged at the bottom.

This furnace had two distinct improvements over the original Parkes furnace: It had six hearths instead of two, and they were so thinly constructed that the arch of each heated the floor immediately above it, which enabled it to run on the heat supplied by the combustion of the pyrite. Again, the ore traveled constantly from the center to the circumference of one hearth, and reversed the direction upon the next one, thus making the operation continuous and the roasting more regular. The repairs, however, were heavy and, as the arms were attached rigidly to the vertical shaft, it became necessary to shut down and cool the entire furnace whenever they, or their plows, required repairs.

This may be called the No. 1 McDougall roaster.

Herreshoff simplified, enlarged, and improved greatly the entire construction of the shell and of the interior mechanism, rendering the furnace peculiarly suited for the thorough roasting demanded in the manufacture of sulphuric acid from pyrite. He substituted a light, hollow, air-cooled shaft for the heavy solid 8-inch shaft of the original furnace, and fastened his rabble-arms to the shaft by a simple locking device which permits of their ready removal without cooling the furnace. The outside diameter of his shell is eleven feet, and the thickness of its red brick lining is eight inches.

This may be called the No. 2 McDougall roaster.

The No. 3 type of McDougall's furnace is the modified Evans-Klepetko, introduced in Montana in 1892. Its diameter is 16 feet or more, its arms are water-cooled, and its capacity is astonishing — as high as 40 tons (36 metric tons) daily of ore containing upwards of 35 per cent sulphur being roasted down to seven per cent sulphur, without fuel. As this is the type of furnace selected by several of the greatest copper plants of the world as best adapted for the imperfect roasting demanded by the copper smelter, I shall describe it in detail after a rapid review of the remaining important types of roasters.

Years before the McDougall furnace had been applied to any extent to the roasting of copper ores, O'Harra, in 1885, built in Butte, Montana, a mechanically stirred furnace consisting of two long, narrow hearths, one above the other, heated by several fireplaces situated at intervals along their sides. Gangs of small plows, dragged by an endless chain, made a complete circuit, passing through the upper hearth, out into the air to cool, and back through the lower hearth, the ore being gradually moved ahead until it was discharged roasted. The capacity was great and the operation continuous, the O'Harra furnace roasting about 61 lb. ore per square foot hearth area per 24 hours (298 kilos per square meter) instead of 25 to 35 lb., which was the duty of the hand furnaces. Certain mechanical defects — some remediable, others inherent — caused so much delay and expense as practically to neutralize the great advantages of this apparatus. This furnace has been modified and improved by Allen and by Brown. The accompanying illustration shows the Allen-O'Harra furnace in which the gangs of plows are mounted upon wheeled carriages running upon tracks laid upon the floor of the hearth.

The ore is fed automatically from the hopper A to the upper hearth B, and is gradually moved by the plows toward the further end of the hearth, where it drops through the slot C to the lower hearth D. It thence traverses the lower hearth until it reaches the discharge E. The chain is driven by the sprocket-heel F, on the shaft G, and is kept taut by the wheel H in the sliding frame I, which is provided with a weight, J. Six sets of plows, K, are attached at equal intervals to the chain. They are carried on wheels running on the track L. The chain is also supported by simple trucks M, midway between the plow-carriages. It will be noticed that the vanes on the separate halves of the same plow turn furrows in opposite directions; also, that the same plow on the upper floor turns furrows in a direction opposite to its furrows on the lower floor, and that each plow turns furrows in a direction contrary to those made by the plow preceding it. A vane set to turn a furrow toward a guide-rail, or wheel, is fastened to the plow-shaft at some distance to the rail and wheel, so as not to cover the rail, nor to throw ore into the path of the wheel. A vane set to turn a furrow away from a guide-rail, or wheel, is fastened on the plow-shaft close to the rail and wheel, so as to turn

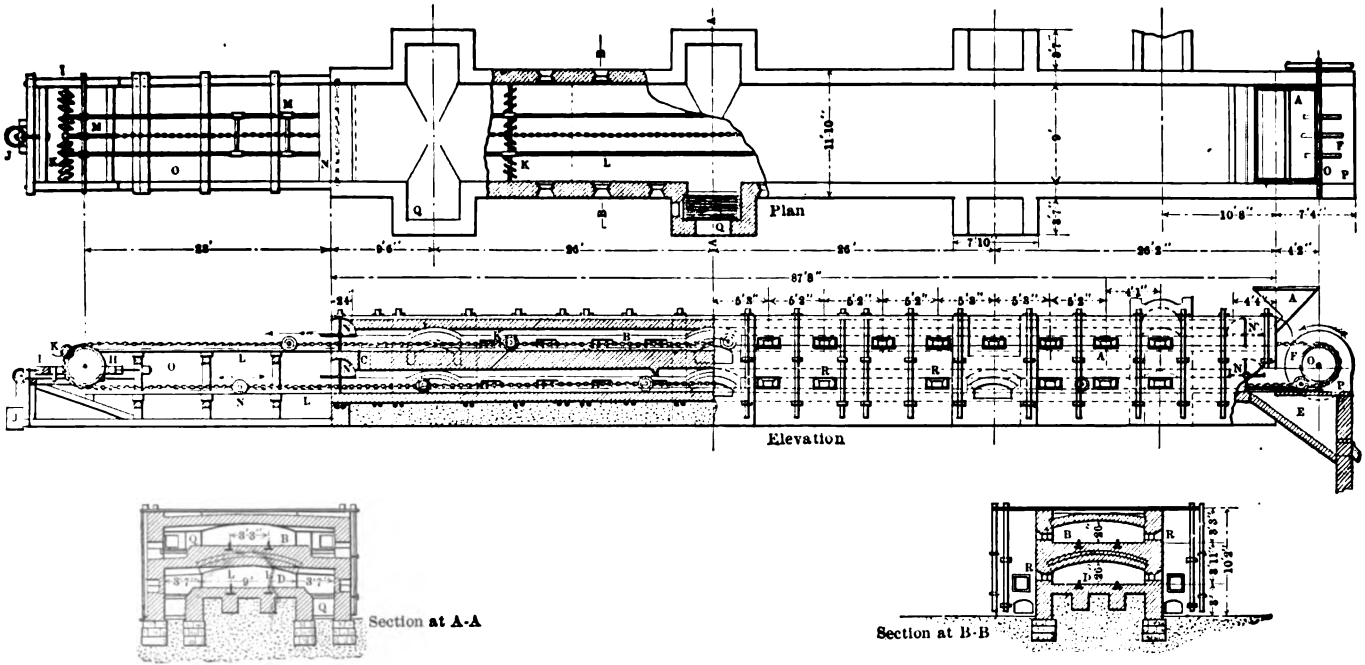


FIG. 9. — The Allen-O'Harra calciner

the ore away from them. The arrangement of the vanes on the separate halves of the same plow, by which they turn furrows in opposite directions, balances the tendency of the plow to be forced off the track on the side opposite to the direction of the furrows, which it would have if the furrows were all turned in the same direction. The hearths are closed at each end by horizontal turnstile doors N, actuated by the moving carriages. The cooling space O for chain and plows is 23 feet in length. The grid P at the driving-end of the furnace is intended for convenience in repairing chain and plows. There are five pairs of fire-boxes, three for the lower hearth and two for the upper, though only one or two pairs are commonly used. The doors R are provided with dampers to admit air to the hearth. The tie-rods that pass through the upper and lower floors are protected by 2-inch pipes, and may thus be easily renewed if burned out.

Brown modified the Allen-O'Harra furnace by partitioning off a narrow corridor on either side of the hearth, in which he placed his tracks. The carriages, attached to an endless wire rope — like a cable-car — ran in these comparatively cool corridors and supported a rigid arm which projected, at a right angle, into the hearth, and to which the stirring plows were attached. The partition between corridor and hearth was formed by a wall built up from the hearth for a certain distance, and by tiles projecting downward from the arch so far as nearly to meet it, thus leaving a continuous, narrow slot, along which the rabble-arm traveled. This continuous slot of Brown's became one of the most essential features in this type of roaster, and has been the subject of extensive litigation.

By means of guiding-pulleys, a wire rope may be made to travel in any desired direction, and the attached plows must follow wherever it conducts them. Consequently, Brown built his furnace in the form of a circle, an ellipse, or any other shape suited to the exigencies of the situation. The accompanying illustration shows the type of his furnace which is used very successfully in roasting blende ores at the zinc smelters of the middle West.

The annular hearth A is broken at B for the ore-discharge, and to afford a cooling space for the plows. These are not shown in the drawing, but are mounted on wheels running upon rails in the lateral galleries C and D, Figs. 1 and 2. The inner rail *c* is of

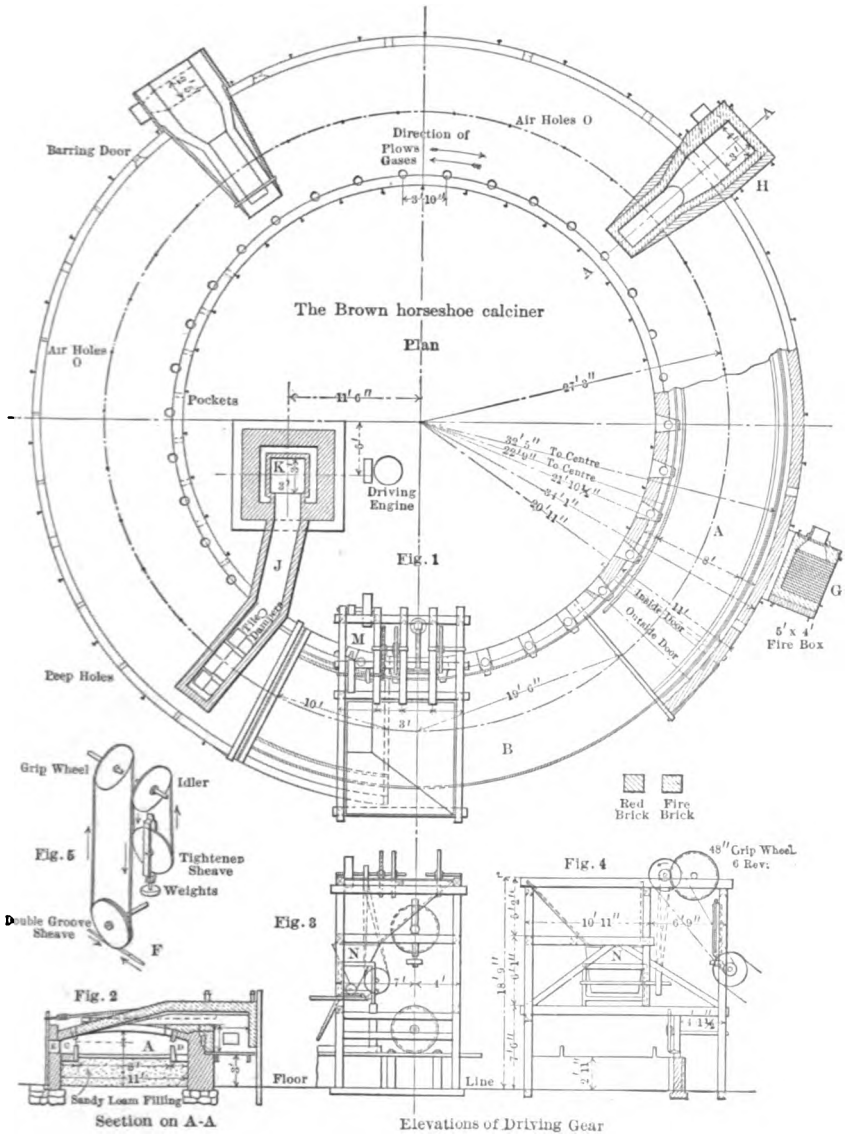


FIG. 10. — The Brown horseshoe calciner

iron; the outer one *d*, of hard-baked tile, except in the broken portion of the furnace. The plow carriages are moved by an endless cable F, Fig. 5, which runs around the little horizontal rollers E, and is driven by the ordinary cable-car mechanism, shown in perspective in Fig. 5. The flame from the three fire-boxes G, H, and I enters the hearth and passes out through the flue J into the stack *k*. The heated plow, which had just completed the circuit of the furnace, comes into the open air at L. It soon comes in contact with the cooled carriage that has been standing in the open for some minutes, and pushes it ahead to where it is gripped by the cable at M, the heated carriage remaining in the place of the cooled one. The ore is fed from the hopper N, and is discharged at H. The rollers E, which are mostly outside of the hearth (see Fig. 2), the cable F, and the rail *c*, are so cooled by the external air and inward draught as never to reach a temperature of 150 deg. Fahr. (65 deg. Cent.). Air is also admitted through the roof by means of the holes O. As there are no revolving arms, the hearth is braced with tie-rods in the usual manner.

About one and one-half horse-power is required to run the machinery. In roasting heavy zinc-blende ores, about 20,000 pounds of finished product is made per 24 hours, the ore averaging over 30 per cent sulphur, and being roasted down to 0.85 of one per cent.

The Wethey Furnace. — The Wethey furnace has the same long rectangular hearth (usually four superimposed) common to the class of furnace now under consideration. It has, however, an important distinctive feature which consists in the method of supporting the roof-arch, and providing a cool track for the wheels of the carriage which bears the plows. The skewbacks of the arch are I-beams hung from horizontal transverse I-beams, and held in place by short struts riveted to the I-beam buckstaves.

This method of construction prevents distortion of the structure and renders it possible to arrange a slot the entire length of each side wall, so that the main portion of the plow-carriage runs upon a track in the open air. The slot is opened and closed automatically as the carriage passes along.

The Ropp Furnace. — The Ropp "straight-line furnace" has the ordinary long flat hearth of this type of reverberatory roaster, through which a continuous cable draws a series of plows which move the ore steadily from the feed to the discharge end in the

usual manner. Fire-boxes on either side of the long hearth maintain the required temperature.

Underneath the middle line of the hearth passes a longitudinal tunnel in which is laid a track, and upon this track runs a series of trucks attached at suitable intervals to the endless cable. Each truck has a vertical arm which protrudes through a slot traversing the center line of the hearth, and which carries the rake arms with their attached plows. After traversing the hearth, the track turns 180°, so that the plows travel back to their starting-place in the open air, and are thus cooled. A 14 × 150 ft. furnace (inside measurement) may cost to build somewhere about ten thousand dollars. It requires about 95,000 lb. of iron and steel (43,200 kilos), 175,000 red brick, and 11,000 fire-brick.[1]

The Keller-Gaylord-Cole roasting furnace consists of two sets of six superimposed hearths, the driving mechanism being situated between these two blocks, and the entire structure constituting a single furnace.

There are, of course, six sets of rabble-arms on each side, one set above the other, projecting through slots into their respective hearths. The rabble-arms are provided with plows both above and below, and these plows are in contact with the ore only when traveling in one direction. When their motion is reversed, a tripping mechanism turns the arm one-fourth of a revolution, so that both its sets of plow-blades lie horizontally above the ore, and in this position the rabbles move back to the other end of the furnace. When they reach this point, the arm is again tripped and revolves 90 degrees. But the revolution of the arm always continues in the same direction, so that the plows that were at first projecting perpendicularly into the air are now brought into use. By this device the ore is moved always in the desired direction. The two sets of plow-blades are so fitted on the rabble that they constantly alternate in the ridges and furrows of the ore on the hearth.

The driving gear consists of a wire rope, the extremities of which are attached to the rabble-frames, while the ropes themselves pass around a large driving-wheel, on whose shaft is keyed a pinion that receives reciprocal motion from a rack actuated by a hydraulic piston.

The slots are closed by traveling steel tapes, as in the turret calciner, but this furnace being longitudinal, and the motion of

[1] Ingalls' "The Metallurgy of Zinc and Cadmium."

the rabbles being reciprocally to and fro, the tapes are wound and unwound alternately on horizontal pulleys, placed at each end of the hearth. These are governed by springs so as to keep the tape taut, and its winding is assisted by counter weights.

The hearths are three feet apart vertically, and are covered with silicious tailings from the concentrator. The enormous mass of brickwork contained in the superincumbent hearths and arches retains much of the heat generated by the oxidation of the sulphides, and consequently diminishes the fuel consumption to a point that would seem impossible to those who have not given attention to this particular subject.

There is a $2\frac{1}{2}$ x 4-foot fireplace, fired with slack coal, to each block of hearths; that is to say, two fireplaces to the double block forming a single furnace. The flame is only allowed to traverse the top hearth, where it is used to ignite the sulphur quickly, the temperature on the lower hearths being ample without extraneous heat to reduce the sulphur to the required standard — 7 to 10 per cent.

The Pearce Turret furnace has long been known as one of the best designed of that class of furnaces in which the ore is stirred by means of plows attached to radial arms, and thus in a circular moving path.

It is peculiarly adapted to the tolerably thorough roasting of difficult ores, and especially of such as cannot maintain the required temperature by their own combustion. The modern copper smelter, who desires only imperfect roasting, and who intends to effect this roasting without the aid of carbonaceous fuel, sees himself more and more driven to the employment of furnaces which are designed especially to conserve the heat liberated by the oxidation of his sulphides.

The accompanying drawings illustrate one of the older Pearce furnaces, no radical changes in them having been made, except — in some cases — to place several hearths one above the other.

A is the hearth, forming a circle with a wedge-shaped piece removed at B, for the discharge of the roasted ore. This hearth is constructed over the dust-chamber C, through which the gases pass in a direction contrary to that in which they move upon the hearth. D is the first fireplace and E the second one, the gases moving around the hearth to the flue and downtake F, through

which they pass to the dust-chamber. The inner hearth-wall has a continuous slot G for the passage of the spoke-like rabble-arms, H, which have their hub J around the central column I. This column is stationary, and is hollow to admit of the passage of a

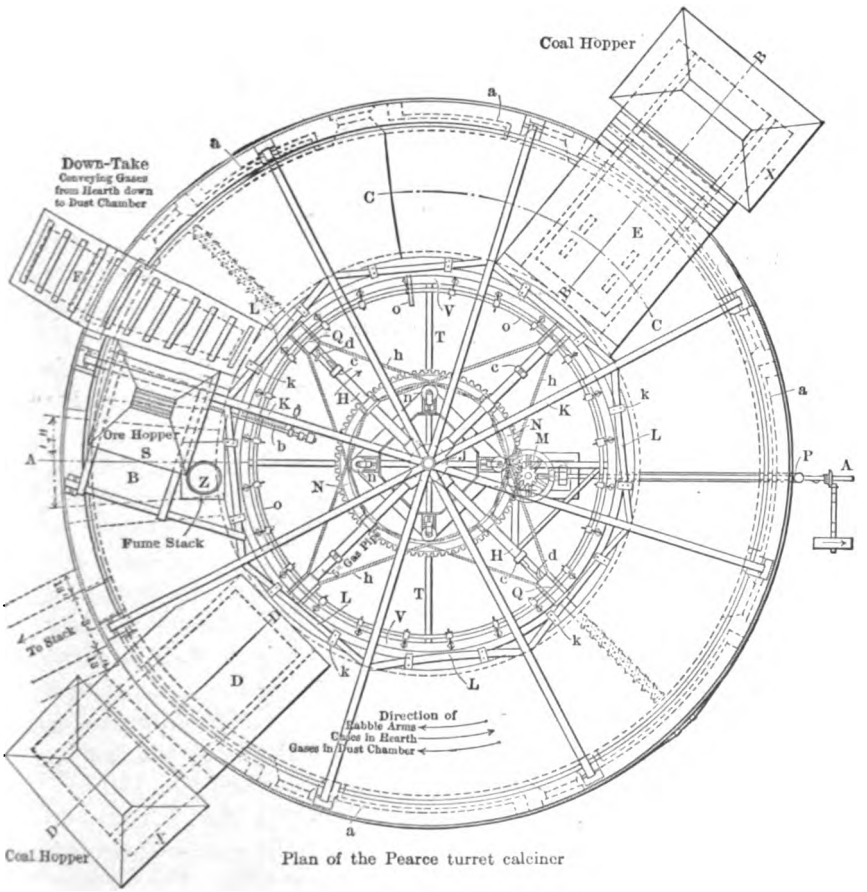


FIG. 11. — Pearce turret furnace — plan

light blast of air to the wind-box (hub) J. The superior portion of the inner wall and skewback cannot be built up in the usual manner, and is therefore hung from the eight 12-inch I-beams K by means of stirrups *k*, and the cross-beams L. The rabble-arms H are strongly braced by means of the straining rods *h*, and are

revolved by the pinion M which meshes into the bull-wheel N. This wheel is centered by the rollers *n*, and the entire weight of the rabble-arms and driving gear is taken by the conical rollers O running on the circular track *o;* no weight at all comes upon the hub J. A 5-inch pipe P protects the driving-shaft *p* where it traverses the dust-chamber. The rabble-arms have a joint at Q so that they can be adjusted to suit the wearing of the plow-blades. The blast coming through the pipe R, the central column I, and the wind-box J continues through the rabble-arms H (which consist of 5-inch gas-pipe), and, cooling that portion of the arms which is exposed to the heat of the gases, streams out into the hearth through the openings *h* and the little pipes *h''*, thus cooling the plows and furnishing hot air for the oxidation of the ore. On the first portion of the hearth where the fresh ore is being gradually heated, no air is desired. The blast is, therefore, cut off opposite the ore hopper by means of the butterfly valves *aa* (Figs. 1 and 2), which are closed by the stop *b*, and again opened at *c*. Heated air is also introduced through the exterior wall of the hearth by means of the intramural passages *d*.

The ore is dropped upon the hearth from the hopper S by an automatic feed mechanism actuated by the stops *e* on the rabble-arms. It is gradually advanced by the plows in a direction opposite to the gases, until it is discharged at B into a car. The 12-inch I-beams K take their bearings on the central column and on the main outside wall of the furnace. This calciner is strongly banded externally, and is internally braced by the 6-inch struts T that radiate from the central column. The slot G is closed by a 12-inch steel tape U that revolves with the rabble-arms, and is supported and pressed outward against the walls of the slot by means of the bell-cranks and weights *u*, Figs. 8 and 9. The bell-cranks are supported on a circular angle iron V that is bolted to the rabble-arms. The fire-boxes burn slack coal, and are provided with a step grate W and automatic coal hoppers X. The fireplace E, nearest the feed, is provided with a curtain arch Y, Fig. 3, as the ore is easily fusible at this stage of the roasting. There are four rabble-arms, but it is found best to use only two of them. The discharge vault is provided with a light stack Z to carry off the fumes.

About two horse-power is required to run the furnace and blast. Apart from repairs and renewals, which are slight, no

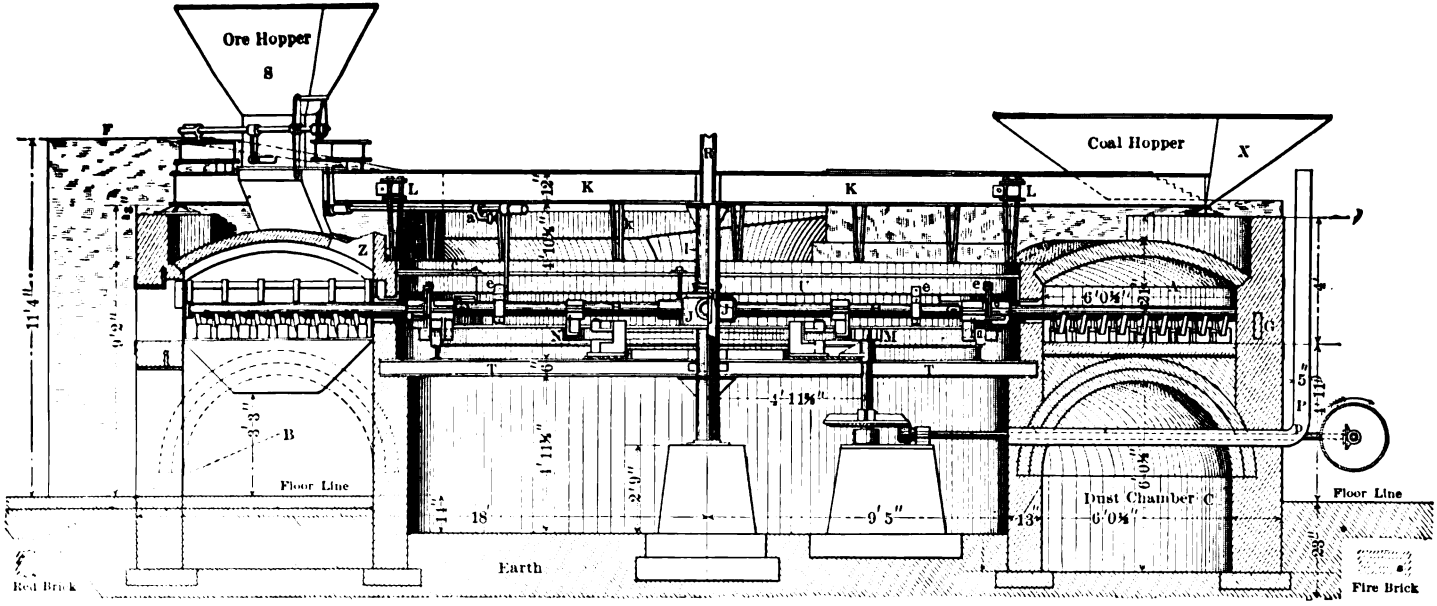


FIG. 12. — Pearce turret furnace — section

labor is required at the furnace except to oil the machinery, to fire, and to have a general supervision of its behavior.

Some of the results obtained in ordinary work by this furnace are as follows:

Of iron pyrites containing 43 per cent sulphur and crushed to pass a two-mesh screen (9 mm. openings), 16 tons per 24 hours are roasted to 6 or 7 per cent sulphur, using $2\frac{3}{4}$ tons of Colorado slack coal.

Of matte from the lead smelters, containing 11 per cent lead, 15 per cent copper, and 17 per cent sulphur, crushed through a six-mesh screen (3 mm. openings), 11 tons are roasted in 24 hours to 3.3 per cent sulphur.

Of concentrated stamp-mill tailings (pyrites), with 45 per cent sulphur, and 10 per cent silica, 9 tons were dead-roasted in 24 hours, to show the utility of the furnace for roasting for the extraction of gold by chlorination. No trace of sulphur remained in the roast.

Of Butte concentrates from the Gagnon mine, consisting of variable mixtures of pyrites and blende, but always high in zinc and sulphur, 15 tons per 24 hours are roasted to 6 or 7 per cent sulphur. The following analysis represents an average sample of these concentrates:

silica	18.2	per	cent
iron	20.3	"	"
zinc	14.85	"	"
copper	11.29	"	"
sulphur	31.53	"	"
Total	96.17	"	"

There are various other excellent roasting furnaces, but those already enumerated are the especial types that have been designed for the treatment of copper ores, and the limits of this work forbid further description.

The McDougall Furnace. — This furnace, as now employed, is the legitimate outgrowth of Parkes' original automatic roaster, improved by McDougall, and enlarged and adapted to modern requirements by Evans, Klepetko, Herreshoff, and others. Introduced into Montana in 1892 by Klepetko for the roasting of the pyritous Butte concentrates, its worth was soon appreciated, and it is now the leading roasting furnace in the United States for the treatment of material of the above description.

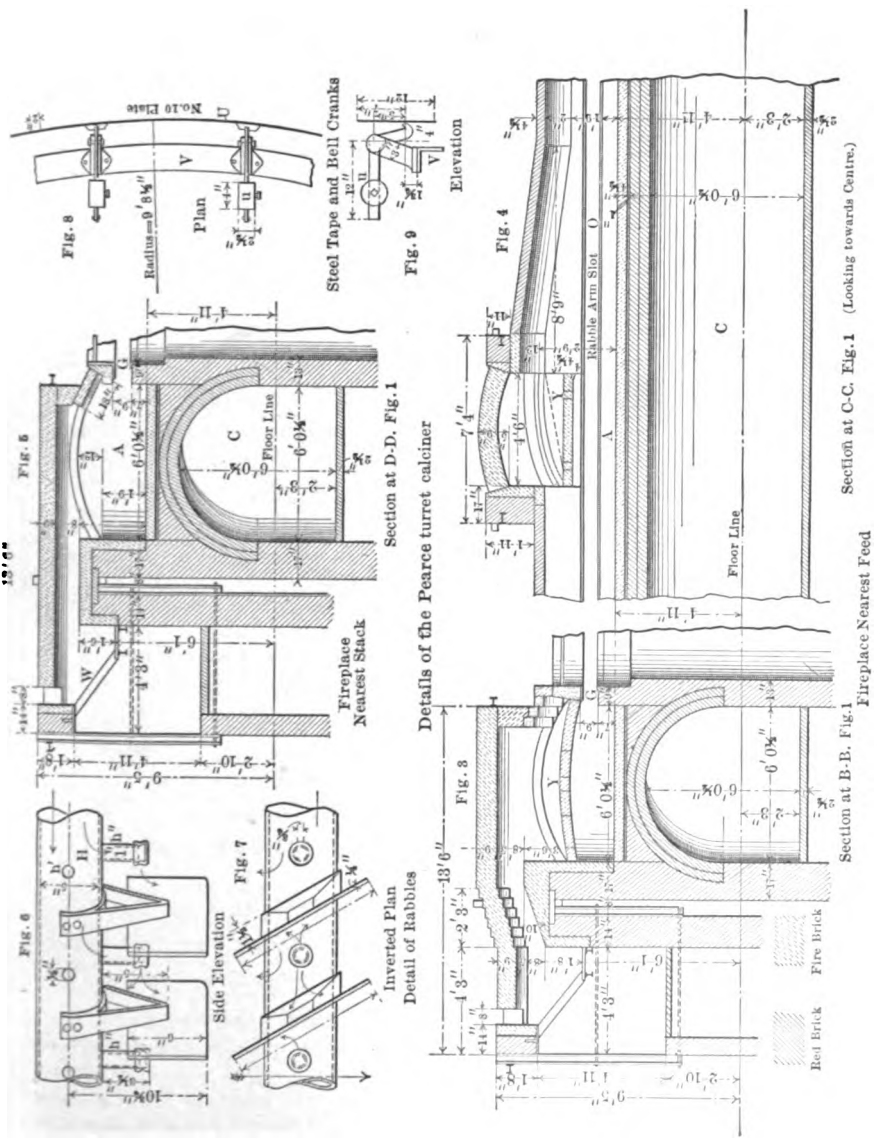


FIG. 13. — Pearce turret furnace — details

For this reason I have deferred the discussion of the actual *practice* of roasting until I could take it up in connection with the description of the apparatus most largely used.[1]

It must be borne in mind that the succeeding description and discussion refer to the McDougall furnace as designed for treating heavy pyritous ores, without carbonaceous fuel, and with the intention of leaving seven or eight per cent of sulphur in the roasted product. As this is the normal and ordinary situation in which the copper smelter would employ a roasting furnace at all, it seems proper to concentrate our study accordingly.

The McDougall roasting furnace is a mechanically operated circular, multiple-hearth furnace of the self-contained type, having five or more superimposed hearths. The general plan of the furnace consists of a cylindrical shell made of tank-steel plate, which contains and supports the various hearths, shafts, rakes, feeder, etc. This shell, together with the cast-iron plates which form the bottom of the furnace, is, in the most modern type of furnace, supported some 10 or 12 feet above the ground level by structural steel beams and columns; the latter also carry the driving machinery.

The shell of this furnace, when used on self-roasting ores, has a nine-inch brick lining from which are sprung, at intervals of three feet, flat arches which form the various hearths of the furnace. These arches are also nine inches in thickness, the entire brickwork of the furnace consisting sometimes of ordinary red brick. It is, however, more advantageous to use fire-brick around the drop-openings, and at other points which are exposed peculiarly to mechanical wear.

The ores are fed by an automatic appliance through the roof, or top arch, of the furnace, in such a manner as to seal this opening against the entrance of cold air, or the egress of gases. The

[1] Much of this description is based upon personal observation at the Great Falls and the Washoe smelters in Montana. The former plant has 22, and the latter 64 of these McDougall roasters, and the liberal attitude which these great corporations assume toward the scientific investigator, the freedom with which all details of the work are opened to his inspection, and the friendly coöperation of their research departments have made it possible to give minute and reliable information concerning the McDougall roasting, covering long periods of time and unusually large quantities of material treated.

mechanical arrangement for stirring the ore, and for conveying it from one hearth to the next one below, consists of a vertical, hollow, cast-iron shaft extending through the several hearths of the furnace, having two or more radial cast-iron arms attached to it for each hearth. Suspended from the under side of these arms are cast-iron plows or rakes, which are set at an angle to their direction of rotation, so that their effect is not only to stir continuously the material under treatment, but also to force it gradually towards the center, or the periphery, of the respective hearths, where there are openings through which the ore is discharged to the hearth next succeeding. The bottom hearth discharges its finished product into a steel bin, from which it is drawn into cars to be conveyed to the smelting furnaces.

The central shaft and the arms are provided with a very satisfactory water-cooling system, which prevents the sagging of the arms, and the long list of troubles arising therefrom, and consequently permits the use of large hearths.

The power required to operate a single furnace varies from one and one-half to four horse-power, dependent upon conditions which will be considered later. The driving machinery, as commonly arranged, is belted from a countershaft attached to the furnace columns, or to the side of the building. A friction clutch is provided, which may be located upon this countershaft or upon the driving shaft of the furnace immediately underneath it. The operating lever of this clutch is connected to a vertical shaft which extends upwards through the several floors (usually three) from which the furnace is worked, enabling the operator to stop and start the stirring mechanism from any floor. The small amount of power required to operate the automatic feeder is taken from the top of the vertical shaft, which extends through the top hearth for the attachment of gearing and for connection with the water-cooling attachment.

The McDougall furnace is built in different sizes, and with certain variable features. Some of these furnaces have small fire-boxes (enclosed or separate) at the bottom, to assist in starting the furnace or to supply extra heat for ores low in sulphur. Some are built upon masonry foundations, while others are supported by a skeleton structure of steel beams and plates, etc. These details vary to suit individual conditions, and are not fundamental either in principle or in practice. The accompanying cuts illus-

trate different types of the furnace, but contain the essential features common to all the types.[1]

Fig. 14 is an elevation and vertical section of one of the larger standard McDougall roasters, having an outside diameter of 18 ft. 5 in. (5.6 m.), and containing six hearths, with an enclosed fire-box under the sixth (lowest) hearth.

Fig. 15 is an end elevation, and Fig. 16 a side elevation and partial vertical section of the standard McDougall roaster of 18 ft. (5.5 m.) outside diameter, arranged with structural steel

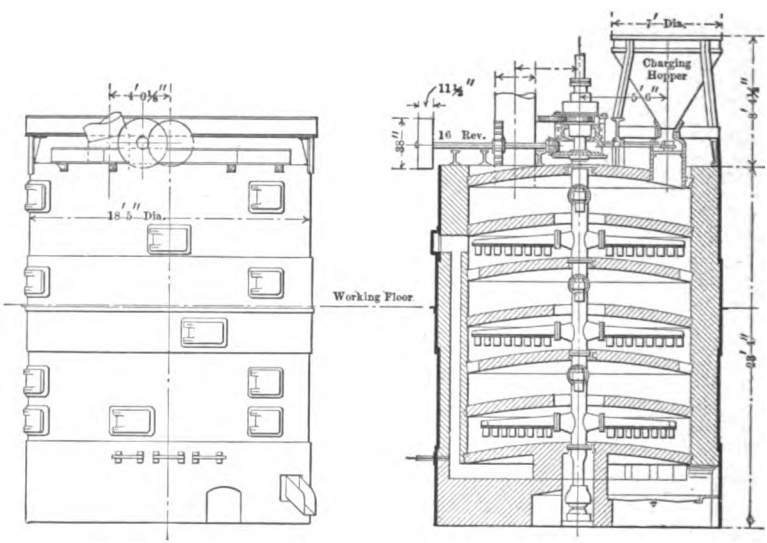


FIG. 14. — McDougall calciner

supporting-frame, and hoppers for the roasted product. The weight of the entire ironwork for such a furnace is about 90,000 lb. (40,824 kg.). There are required for its construction about 37,000 red brick, and 500 fire-brick.

The Washoe smelter of the Anaconda Copper Mining Company contains the largest roasting plant in the world, so far as I am aware. This plant consists of 64 McDougall furnaces contained in a single steel building, 96 × 412 ft. (29.3 × 125.6 m.). The system of roasting here prevailing is the outcome of a vast amount

[1] I am indebted to the Allis-Chalmers Company for these cuts, as well as for a portion of the description of the mechanical details.

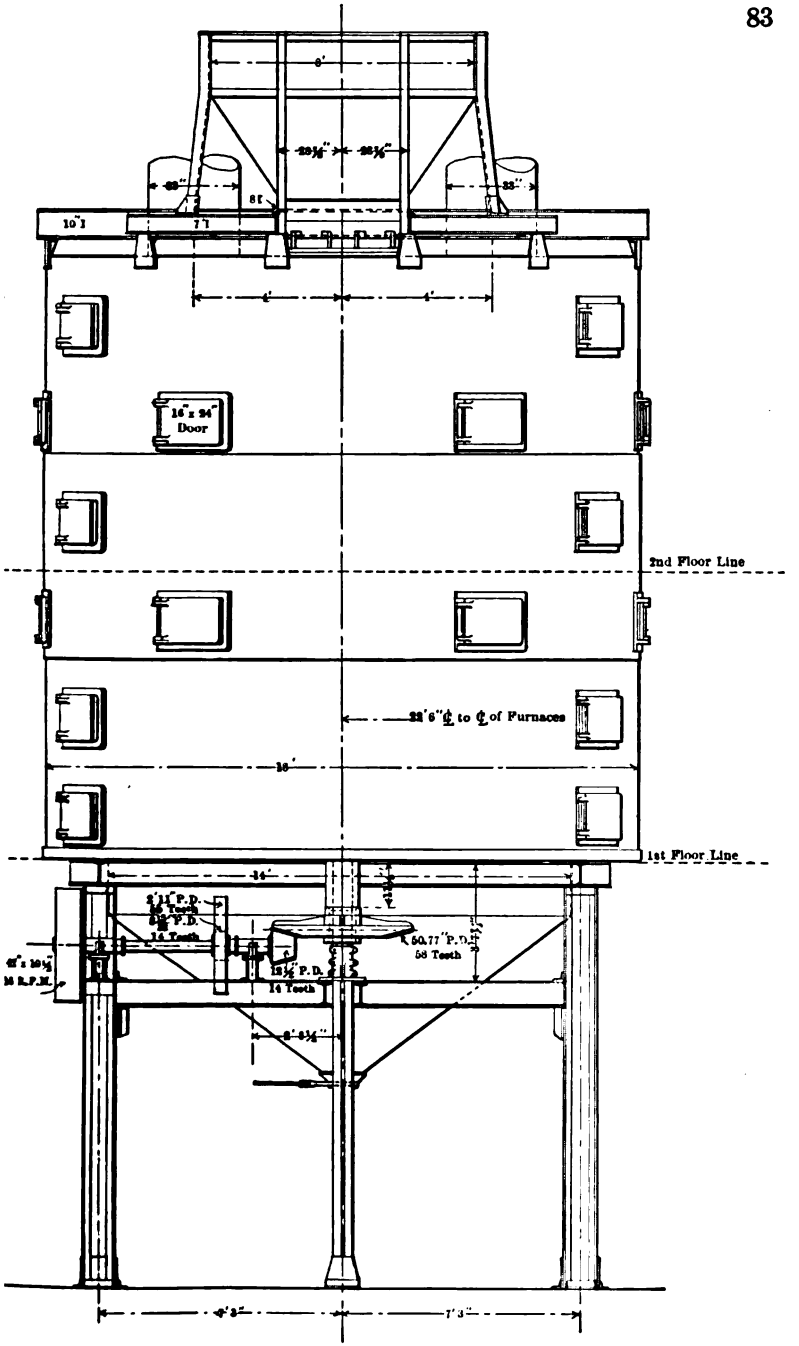


FIG. 15. — McDougall calciner

of practice, thought, and research, and I feel that it is more useful to occupy considerable space with a thorough description of the best modern practice than it would be to attempt to review in a more superficial manner a greater number of plants.

The 64 McDougall roasters of the Washoe smelter are placed in 16 rows of four in a row across the building, each row of four receiving its power from a single main shaft, with which the individual furnace is connected by a friction clutch. The main working platforms are common to each two rows; consequently, there are eight blocks, each block containing eight furnaces. The platform structure is of skeleton steel, and is three stories high — two hearths being worked from each story.

The furnaces are 18 ft. 3 in. (5.55 m.) high, and 16 ft. (4.87 m.) in diameter outside. They are placed 18 ft. (5.5 m.) from center to center in one direction, and 21 ft. 3 in. (6.48 m.) in the other. They are enclosed in a boiler-iron shell $\frac{3}{8}$ in. (9.5 mm.) thick, and are lined with a full course of red brick, leaving an inside hearth diameter of 14 ft. 6 in. (4.42 m.). There are two gas flues 24 in. (0.61 m.) in diameter to each furnace, the flues from each block of four furnaces connecting with a brick cross-flue having openings for discharging the flue-dust into pipes which empty into special steel receiving hoppers below, whence it is removed in cars either to be re-roasted, or to be charged in limited amounts to the reverberatory furnaces.

The central cast-iron shaft has an inside diameter of nine inches, and is made in three sections bolted together, with the interposition of copper gaskets. At each hearth a pair of hollow cast-iron arms is bolted to the shaft, tight joints being secured by the use of copper gaskets.[1]

The rakes (blades, plows) are of various shapes and sizes, and have different devices for easy and quick attachment to the stirring-arm. They are usually of cast-iron, made in two pieces, and keyed

[1] Formerly, the cooling water was brought down through the central column in a three-inch pipe — provided with a stuffing-box — from which it was distributed to the outer ends of the stirring arms by horizontal one-inch (25 mm.) pipes, the heated water passing back through the hollow arms to the central column, and rising to the top where it is discharged into a stationary launder. It is now found more convenient to omit entirely the one-inch horizontal pipes and to discharge the cold water from the three-inch pipe at a point near the bottom of the hollow central column. As the water becomes displaced by constant accession from below, it rises slowly

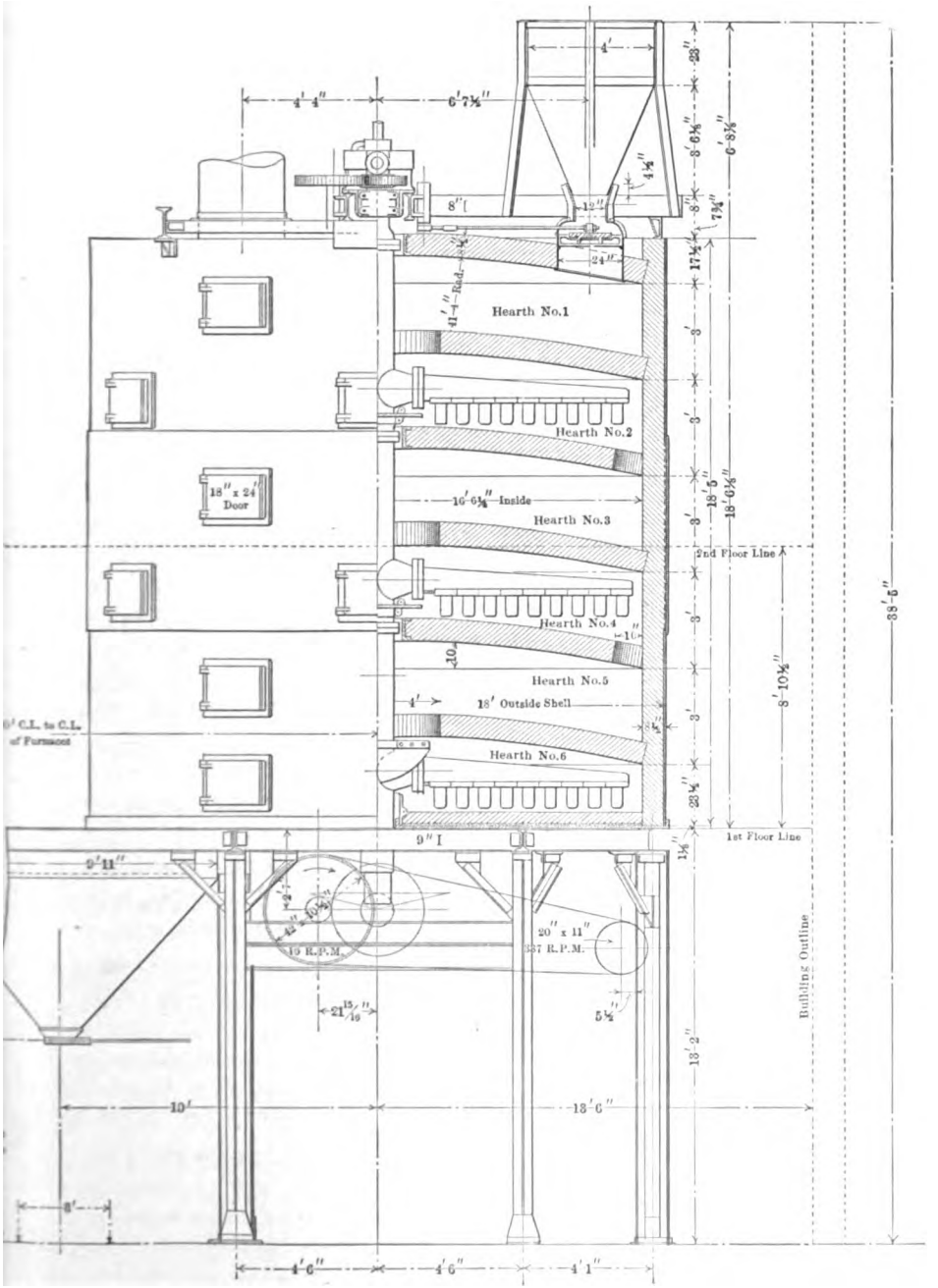


FIG. 16. — McDougall calciner

to the arm by a wedge-shaped blade fitting into a slot. It is important that they should be capable of being replaced quickly and independently.

Each furnace requires about 20 gallons (76 lit.) of cooling water per minute to maintain the overflow temperature at 80° C.

Each block of furnaces is run by an independent electric motor and, while plenty of surplus power is provided, careful tests have shown that a furnace running under normal conditions uses only about 1.7 H.P.

The older McDougalls were elevated upon a high masonry foundation to obtain the necessary position for the discharge of their product direct into steel hoppers, to be drawn off into cars as required. The more modern construction is to place the furnace upon an elevated framework of structural steel, this system presenting advantages that are obvious.[1]

The erecting of a self-contained mechanical roasting furnace presents no difficulty to the experienced builder; but there are always two points which demand more than ordinary care. One of these is to provide an ample and solid foundation; while the other — and more complicated one — is to build the hearth arches in such a manner that they shall be firm and solid, and yet when heated shall not rise sufficiently to interfere with the plows which sweep continuously above their upper surfaces. While this work should always be executed under the supervision of an experienced furnace mason, it may be useful to consider some of its more salient features.

The eight-inch red brick lining of the main shell presents no difficulties; but the construction of the six circular flat arches which form the roof of one hearth and the floor of the next one

and flows along the horizontal bottom of the hollow in the arms, returning to the central column along the upper surface of the arm-hollow. This return is facilitated by shaping the hollow in the arm in such a manner that its floor shall be horizontal, while its roof shall be inclined — rising steadily from the peripheral end, and attaining its maximum height at its point of juncture with the column. The arms on the top hearth have no cooling cavity.

[1] The only criticism that I have heard the McDougall workmen themselves make upon this latter method of construction is that there is so great a radiation of heat from the bottom of the furnace as to hamper any work at the gearing or driving mechanism to a serious extent. This is a detail of construction that can, of course, be easily remedied.

above is complicated by several difficulties. In the first place, there must be a large central opening for the main vertical shaft which carries the stirring-arms, and the brick at the periphery of this opening must be fastened in such a manner that neither wear nor expansion and contraction shall loosen them. There must also be drop-holes from each hearth to the next lower floor, and these are arranged alternately at the central opening, and at the extreme periphery of the hearths, and protected by iron castings.

The floors are numbered from the top down. The odd-numbered hearths have their drop-holes at the center, while the even-numbered floors have the drop-holes at the periphery, the plows being set at such an angle that the ore travels gradually backward and forward across alternate hearths until it is discharged into hoppers through the two peripheral openings of the sixth, or lowest, hearth.

The first (top) hearth has one drop-hole and three doors.
The second hearth has six drop-holes and six doors.
The third hearth has one drop-hole and three doors.
The fourth hearth has six drop-holes and six doors.
The fifth hearth has one drop-hole and three doors.
The sixth hearth has two drop-holes and four doors.

Allowance for the expansion and contraction of the brickwork must also be made, so that the entire hearth, when heated, may not rise enough to interfere with the plows which sweep over its surface, nor, when cooled, contract sufficiently to fall through bodily. While these points are familiar to all furnace masons, the best manner of handling them in any individual case can be attained only by years of experience in building and repairing the especial type of furnace under consideration. Thus the practice of construction and repair at Anaconda will be useful.

The hearths which have their drop-hole in the center may be laid entirely of red brick, the 14.5 ft. hearth (4.42 m.) of the Washoe furnace requiring 1350 red brick of ordinary shape, 600 side-wedge, and 300 end-wedge — total, 2250 red brick for the one floor. For the hearths with six peripheral drop-holes, it is better to edge the openings with fire-brick, as they are subjected to constant mechanical wear as well as occasional barring. Such a hearth will require of red brick: 1200 of ordinary shape, 600 side-wedge, 300 end-wedge; and of fire-brick: 40 of ordinary shape, 140 side-wedge, 20 end-wedge, 20 splits; making a total of

2100 red brick and 220 fire-brick for the hearth with peripheral openings.

The hearth with central opening is also provided with a central cast-iron ring, cut in halves. This ring encircles the central shaft, leaving an annular clearance space of three inches (0.076 m.) all around the latter. The brickwork of the hearth abuts against the exterior of this ring in its entire circumference, and is keyed into a groove in the ring. The center drop-hole is formed by stopping the brickwork of the hearth so as to leave an annular space of 16 in. (0.406 m.) encircling the vertical shaft. The peripheral drop-holes are 14 in. (0.355 m.) wide on the first and fifth floors, and 18 in. (0.457 m.) wide on the third floor, where there is a strong evolution of sulphur dioxide gas.

The repairs on these furnaces, when run constantly and judiciously, are very light. The shell lining shows little wear after eight years of use. The floors — which are 30 in. (0.76 m.) apart vertically — rise gradually, and require occasional — though infrequent — repair or renewal. Some are still good after seven years' wear; many have lasted for five years. The arch is built originally with a rise of seven and one-half in. (0.19 m.) and a space of three inches (0.076 m.) between the highest point of the floor and the rakes which stir it — the hearth being leveled with crushed limestone of gravel size. If any portion of the arch rises too greatly, it is cut down, or knocked out and replaced with six-inch (0.152 m.) brick.

At the Washoe smelter, one mason and three helpers do the repairing for the 64 McDougall roasters, including their flues and dust-chambers.

Some idea of the time and cost for replacing a single hearth of the McDougall furnace may be gathered from the following details:

The furnace is allowed to cool for two or three days before beginning work. It takes about three days to remove the old floor, construct the arch-pattern, build the new floor, and get ready to fire again. The cost for labor in renewing a hearth averages as follows:

3 days mason's labor at $6	$18
9 days helper's labor at $3	27
labor on arch-pattern	7
Total	$52

Any of the mechanical furnaces are capable of handling material that is far coarser than can be roasted to advantage. Hence it is not profitable to take time to study the maximum size for lumps which might be fed through the furnace without clogging. Considerable quantities of two-inch (0.051 m.) ore have been fed into McDougalls, though with occasional breakage of the plow-blades.

At copper smelters, roasting furnaces are used almost exclusively for the oxidation of heavy sulphides which consist mainly of pyrite and, in a much smaller proportion, of chalcopyrite, or one or more of the richer copper sulphides. As mines seldom furnish direct any considerable quantity of material of this description, it follows that the ore which comes to the roasting department represents only a moderate fraction of the total output of the mine, and that it has been separated from the main bulk of the mine product by some previous operation or process of selection.

As a matter of fact, there are two distinct and separate methods in common use by which this process of selection is carried out.

In the first place, the rich first class ore may be mined by itself so far as possible. Such ore is unfit for any process of mechanical concentration, either because it is too rich in copper to be submitted to so wasteful an operation, or because it is so heavy in sulphides — even though much of them may be only barren pyrite — that it is unfit for concentration, being in fact already concentrated by nature. This ore, when in lump form, is characteristically suited to smelting in the blast-furnace; for its lump form ensures an open ore-column and freedom from flue-dust, while its sulphide content forms a fuel which will go far toward furnishing the heat requisite for its own fusion. But ore cannot be produced strictly in lump form, and the lump ore will always be accompanied by a very considerable amount of *fines;* that is to say, of material whose particles are so small that it will tend to choke a blast-furnace as well as to produce an extravagant amount of flue-dust.

This fine portion of the first class ore is usually richer in both copper and sulphur than the lump ore, owing to the friable nature of sulphides as compared with ordinary gangue-rock; and it is with regret that the metallurgist finds himself compelled to rob his blast-furnace system of this valuable fuel, and to install an expen-

sive roasting plant in which to burn his iron and copper sulphides without experiencing any benefit from the heat thus generated.[1]

As will be seen from the nature of the case, a mere screening of the first class ore suffices to separate it into the two desired portions: namely, lump ore, direct to the blast-furnace; fines, to the roasting department, preparatory to being smelted in reverberatory furnaces.

The division line between the coarse and fine portions of the first class ore is, consequently, a variable one, and may change from time to time at the same smelter. If the remainder of the blast-furnace charge consists largely of coarse material, a finer screen may be used for the first class ore, and finer granules of sulphides may be added to the blast-furnace mixture. If, on the contrary, the ore-column in the blast-furnace is already too dense, or the blast too weak, a coarser screen must be employed, and coarser material must be sent to the roasters.

At the great Butte smelters, a common practice is to pass the first class ore over a one-half inch screen (0.038 m. apertures) as the ore-cars dump their contents into the smelter bins.

The second — and usually far more important — method, by which selection is made of that portion of the ore which must go to the roasting department and be smelted in reverberatories, is the process of mechanical concentration, by which the heavy particles of disseminated sulphides are separated from the greater mass of worthless gangue-rock, and are obtained in a condition of reasonable purity. Unfortunately, the very nature of the operation requires that the entire mass of ore shall be in comparatively fine particles, so that the greater part of the products of concentration, with their enormous store of potential heat, must be sent to the roasting plant in order to fit them for smelting in reverberatory furnaces.

This is a matter of the greatest regret to all copper metallurgists, and, as becomes evident, the problem of to-day is not to see how coarse material our roasting furnaces are able to handle, but rather to find what is the very finest size of sulphides that we can feed into our blast-furnaces without choking them.[2]

[1] One of the most important metallurgical problems of to-day is to perfect a method for the treatment of sulphide fines which may eliminate the roasting plant, and may conserve much of the heat now lost. This subject is discussed in detail on p. 423 et seq.

[2] See p. 429 et seq. for exceptional treatment of raw concentrates, such as briquetting, blast-roasting, etc.

Hence, the material delivered to the roasting furnaces consists usually of first class fines, of concentrates below one-quarter inch mesh (6 mm. aperture), or of a mixture of such products. In Montana, the first class fines consist usually of material which passes through a one-half in. mesh (12 mm. apertures), while the concentrates range in size from what passes a one-quarter-in. mesh (6 mm. apertures) to the excessively fine products of the vanners and slime tables.

In addition to this sulphide material, and to a small amount of returned flue-dust, there is added, at Anaconda, a quantity of limestone equal to five per cent of the sulphide charge. This limestone is to aid in the succeeding smelting operation, and is obtained by passing the coarse limestone flux which is required by the blast-furnaces over a one-in. (25 mm.) grizzly.

It is added before — instead of after — roasting for various reasons, the most weighty of them — in order of importance — being: an intimate mixture of the lime and roasted sulphides is obtained which shortens the smelting operation in the reverberatories; the admixture of the limestone lessens the tendency of the sulphides to soften and form lumps; the carbon dioxide of the limestone is driven off by the excess heat furnished by the roasting process, and thus saves heat during the smelting.[1]

The sulphides and the limestones are dumped from cars into the two tall, 33-ton hoppers with which each of the McDougall roasters is provided. The feeding apparatus operates continuously, and consists of an I-shaped traveler working above slots. It is operated from the main central shaft of the furnace by means of crank and gearing, and feeds the moist concentrates regularly and without trouble, unless too much slime is added. Precautions must also be taken when feeding flue-dust, which runs too freely.[2]

[1] As the roasted ore is fed red hot into the reverberatory smelting furnaces, such free CaO as it may contain has no opportunity to combine again with CO_2 before it reaches the actual smelting operation.

[2] In describing the operating of the McDougall furnaces I have supplemented my own observations at Anaconda by free use of the following papers: "Progress in the Metallurgy of Copper," by L. S. Austin; *Mineral Industry*, Vol. XV. "Notes on the Metallurgy of Copper in Montana," by H. O. Hofman; *Trans. Am. Inst. Mng. Engineers*, Vol. XXXIV. "Washoe Plant of the Anaconda Copper Mining Company," by L. S. Austin; *Trans.* Vol. XXXVII.

The top (first) hearth serves mainly for drying and warming the charge, which contains about six per cent moisture. The temperature of the gases, as they escape into the flue above the furnace, is about 300° C., although the heat varies in all portions of the furnace according to conditions. The gases escaping from this hearth contain about two and one-half per cent of SO_2, by volume.

The plows are so placed in relation to the surface of the hearth that the layer of ore is maintained at a thickness of about three and one-half in. (0.089 m.). The plows wear rapidly on this top hearth, the mechanical effect of the strong decrepitation of the Butte pyrite being more destructive than the chemical effect and the higher temperature of the inferior hearths. They last here only about a month, and as their blades become shorter, a crust builds up on the loose working-bottom. When the plows are to be renewed this crust is removed by slipping a single cutting-plow upon a stirrer-arm and moving it radially towards the central shaft.

The point at which the sulphides begin to burn is variable, depending upon draft, rapidity of stirring, and various other factors. Under perfectly normal conditions, and with the stirrers making one complete revolution per minute, the ore begins to burn upon the second hearth, showing a distinct blue flame by the time the particles have approached the central drop-hole, the temperature being about 600°. On the third hearth the blue flame becomes more intense, flying sparks follow the course of the plows, and the temperature of the flame sometimes reaches 900°, although the heat of the ore is much less. On the fourth hearth the ore has attained an orange red, the sparking has ceased, but oxidation is still brisk, and the temperature still increases somewhat. On the fifth hearth the oxidation begins to wane, as shown by the fact that the discharge temperature is less than the entering temperature. Nevertheless, the maximum temperature is often found in the flame traversing this hearth, reaching 950° in a considerable number of observations. On the sixth (bottom) hearth the ore is still bright, and may have a temperature as high as 850°, cooling to 660° as it falls into the five-ton receiving hoppers. By the time the roasted ore enters the reverberatory furnaces, it has dropped to about 420°. The sulphur is now reduced to about seven and one-half per cent, calculated on the weight of the roasted ore, which is considerably less than that of the raw material. These

temperatures, as well as the accompanying diagram, Fig. 17, are taken from Austin's paper, and run considerably higher than more recent observations.

In this diagram the sulphur content of the raw ore was 32 per cent, and of the roasted material, eight and one-half per cent. The average temperature of the first hearth was about 360°, and that of the finished product in the sixth hearth 870°.

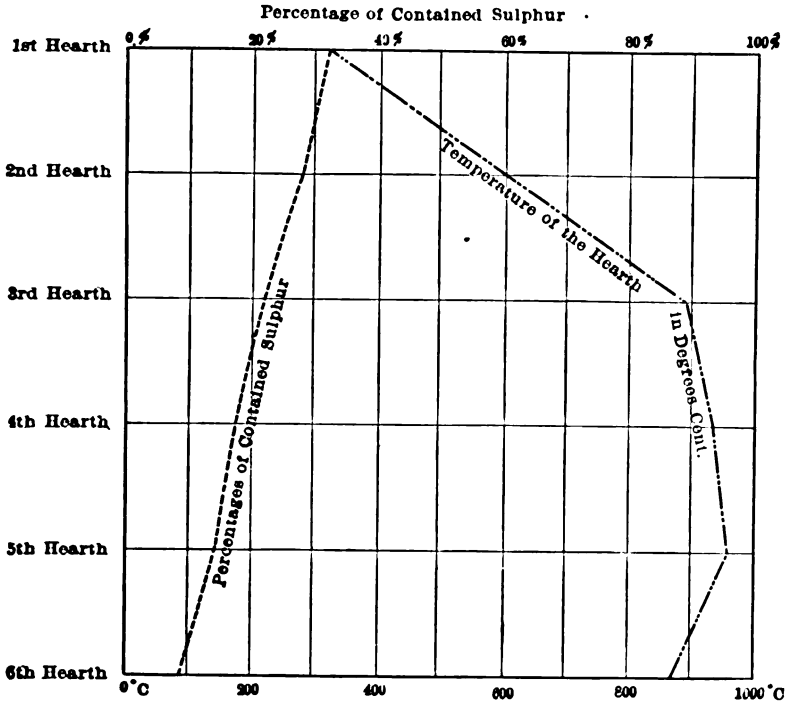


FIG. 17. — Progress of reactions and flame-temperature in the McDougall roaster

As this question of the temperature evolved in roasting pyritic concentrates without extraneous fuel is an important one, I feel that it will be useful to add the results of certain recent observations at the Great Falls and Washoe smelters.[1]

[1] I am indebted to the management of these smelters for the following results, which have peculiar value as they were made by highly trained observers and under conditions which represent very large averages.

The temperatures in these two series of observations average considerably lower than the foregoing ones reported by Austin, due no doubt, in part, to the fact that the table now to be given is based upon temperatures taken by inserting a pyrometer into the peep-holes in the furnace doors.

Average temperatures of 21 Great Falls McDougall roasters taken by Mr. Pyne, and of a considerable number of Washoe McDougall roasters by Mr. Croasdale:

Hearth	Great Falls plant	Washoe plant
No. 1	243° C.	232° C.
2	550	538
3	616	621
4	646	738
5	628	678
6	570	649

It will be noted that the two batteries of roasters have approximately similar temperatures on the first three floors, while the Washoe plant carries a considerably higher temperature during the remainder of the operation.

On May 10, 1909, a series of observations was made at the Washoe smelter upon McDougall roaster No. 42. The furnace was running on ordinary feed — 40 tons (36.3 met. tons) per 24 hours — and with two doors open for admission of air on the lowest hearth, as customary. It appeared to be running a little hotter than usual, as the 12-hour barring-out of the drop-holes had just been completed. A Chatelier pyrometer was used, connected for cold-junction temperature. Each result given is the average of ten observations taken at half-minute intervals.

	Ore	Gases
Uptake	—	371°C
Second hearth, outer edge	610	—
Between second and third hearths	—	738
Third hearth, outer edge	710	—
Fourth hearth, outer edge	821	—
Between fourth and fifth hearths	—	771
Fifth hearth, outer edge	788	—
Sixth hearth, outer edge	705	—

On April 29th similar tests were made on two other furnaces. It is interesting to note the decided variations in temperature

occurring in furnaces which were running satisfactorily and uniformly, and which were all yielding a product containing about seven and one-half per cent sulphur.

	Ore		Gases	
Uptake	—	—	338°	377°
Second hearth, outer edge	443	588	—	—
Between second and third hearths	—	—	638	749
Third hearth, outer edge	538	593	—	—
Fourth hearth, outer edge	732	693	—	—
Between fourth and fifth hearths	—	—	649	705
Fifth hearth, outer edge	732	710	—	—
Sixth hearth, outer edge	616	627	—	—

The length of time required for ore to pass completely through one of these six-hearth furnaces depends upon the speed of the stirrers, the angle of the plow blades, the size of the ore particles, and various other less obvious factors. It ranges, usually, from two and one-half to four hours, being about three and one-quarter hours under the conditions at Anaconda.

A somewhat elaborate experiment was made at the Washoe smelter to determine the relative rapidity of advance of the particles of different sizes. I have been furnished with the following results.

Two furnaces, each treating about 35 tons (31.74 metric tons), dry weight, of ore per 24 hours, and stirred every 35 seconds (one complete revolution every 70 seconds) gave results as follows:

	Size of Particles	
	Between 4.1 and 6.1 mm.	1 and 1.5 mm.
Average time in furnace	165 minutes	190 minutes
Minimum time in furnace	75 "	60 "
Maximum time in furnace	270 "	375 "

There is nothing especial in the operation of roasting sulphides in a self-contained mechanical furnace to distinguish it from the same process when properly conducted in an ordinary long hand-reverberatory roaster.

With a steady draft and uniform ore the process should proceed with regularity; each hearth should maintain its own fixed temperature within moderate bounds; the ignition of the sulphides should begin pretty nearly at the same spot on the same hearth; and the percentage of sulphur in the finished product

should vary only within narrow limits. While this theoretical standard of perfection is never attained in its entirety, there is, in most of the modern plants, a close approximation to it.

This remarkable uniformity of action is aided greatly by two factors, both of which are wanting in the hand-roaster.

The first one is the abolition of carbonaceous fuel as a source of heat. The firing of a roaster with fresh coal produces a sudden depression in temperature, followed soon by a strong rise in the heat of the gases; but even this irregularity is less harmful to the roasting process than the periodical occurrence of the strongly reducing atmosphere which at times accompanies coal-firing.

The other factor to which I shall allude as being an unfortunate accompaniment of the hand-roasting process is the long period of quiescence, with closed doors, high heat, and anything but a strong oxidizing atmosphere — followed by a sudden in-rush of cold air from several working-doors, which remain open during much of the stage of rabbling. If, during the period of quiescence, the heat of the furnace has increased sufficiently so that the hearth can maintain itself at a reasonable temperature during the cooling stage of stirring, it will be so hot that much of the surface pyrites will become sticky and tend to sinter — a circumstance which will have a disastrous effect upon the final percentage of sulphur in the roast. If, on the other hand, the operator avoids this incipient melting of the upper layer of sulphides, there will not be sufficient heat stored within the hearth to outlast the stirring period, and oxidation will be slow and imperfect.

The mechanical furnace, with its steady current of (mostly) preheated air, freedom from the reducing or diluting gases of carbonaceous fuel, and gentle continuous stirring, offers almost ideal conditions for complete and uniform oxidation.

It will be useful to follow the practical operation of the process as carried out at Anaconda and — with slight variations — at all of the smelters which roast pyrite concentrates in McDougall furnaces.

A late average of analyses, extending over a considerable period of time, of the material subjected to the roasting process at Anaconda, gives the following results:

Moisture	8.1	per cent
Cu	7.42	" "
SiO_2	21.2	" "
Fe	26.05	" "
S	33.17	" "
Al_2O_3	2.7	" "
CaO	0.3	" "
	98.94	

There was also present 4.87 ounces silver per ton (0.0167 per cent) and 0.018 ounces gold per ton (0.0000617 per cent).

To this material is added five per cent of its weight in limestone, of which the diameter of the largest pieces does not exceed one inch (25.4 mm.).

The labor at the 64 furnaces consists of three eight-hour shifts and comprises the following persons:

LABOR FOR 64 MCDOUGALL ROASTING FURNACES

	per 24 hours
one foreman per shift	3
one oiler per shift	3
one furnace-man per four furnaces per shift	48
two men loading the calcines into cars per shift	6
four men receiving ore above furnaces per shift	12
two men clearing flues, roofs, gears, etc., one shift	2
one sweeper on charging floor, one shift	1
one sweeper on basement floor, one shift	1
Total	76 men

With an average of 60 furnaces in operation, each roasting 40 tons (metric tons) ore per 24 hours, this gives $\frac{76}{2400}$ of a day's labor to each ton of ore — or, expressed in tons of ore roasted per man-shift, equals 31.58 tons (28. 64 metric tons) roasted per man-shift.

As repairs and renewals constitute a legitimate — though small — portion of the regular labor of roasting, it is proper to add the personnel employed for this purpose, which is as follows:

Repair-staff for 64 McDougall furnaces, flues, etc.
1 head mason
3 helpers
3 repair men (changing rakes, etc.)
3 helpers
1 office boy
Total, 11 men per 24 hours

This makes a grand total of 87 men per 24 hours employed in running 64 McDougall furnaces and keeping them up to a maximum state of efficiency, and gives a duty of $\frac{2400}{87} = 27.586$ tons (25.02 metric tons) per man-shift of eight hours.

Owing to the great size of the plant, its admirable arrangement, the uniformity of its ore-supply, and the ability of its management, this probably represents the extreme of economy attained in roasting material of this description under anything like similar conditions.

The total cost of roasting a ton of ore at these works, inclusive of supplies, renewals, management, and general expenses, is said to be a little less than 30 cents per ton (33 cents per metric ton). With ordinary wages at $3 per shift of 8 hours, this is a very low figure.

The following results cover large quantities of ore treated at this plant. The ore-charge of the McDougall roasters consists of the following products:

screenings from first class ore	3.55	per cent
fine-jig concentrates (the coarse go to the blast-furnaces)	87.08	" "
table concentrates	9.37	" "
	100.00	

To this is added five per cent of limestone, and enough of its own flue-dust to prevent accumulation of this material.

The average amount of ore treated per furnace in 24 hours is 40.53 tons (36.76 metric tons).

1.228 tons raw material produces one ton of calcines. Thus, the loss of weight in roasting is 18.57 per cent.

Assuming that the furnaces make three products — calcines, flue-dust, and barrings — their respective weight per hundred pounds of raw material is:

calcines	78.38 lb.
flue-dust	2.61 "
barrings	0.42 "
Total	81.41 lb.

Average analyses of the calcines (which contain the products of the five per cent of added limestone):

	calcines
Cu	9.32
SiO_2	28.07
FeO	38.42
S	8.01
Al_2O_3	4.38
CaO	3.01
	91.21

Ag per ton..... 4.92 oz. (0.01687 per cent)
Au per ton..... 0.019 oz. (0.0000651 per cent)

The amount of ore roasted per 24 hours per sq. ft. of hearth area is 81.8 lb. (399.14 kg. per sq. m.).

In describing the practical operation of the McDougall roaster it will be simpler to consider the furnace as already heated, and running under normal conditions. The starting of a cold furnace is a comparatively rare and specialized operation, and will be taken up separately.

Referring to the table of labor given on page 97, it will be seen that the duties of many of the workmen are self-explanatory — foremen, oilers, loaders and unloaders, sweepers, etc. — and that it is to the 16 furnace-men per shift that the actual running of the process is confided. Each block of four furnaces is under the charge of one man, but as the working-platforms are so arranged that every two blocks have a common platform, it is customary for the two furnace-men to aid each other in certain portions of the work.

Apart from watching the ore-feed, the cooling water, and the general running of the machinery, the furnace-man pays particular attention to the temperature on the various hearths, stimulated strongly thereto by the constant sampling and analyses of the finished product, where any irregularity in the process is immediately reflected by an increase in its sulphur content.

Under the conditions at Anaconda, the ore — dropped at the circumference of the first hearth, crossing to its center, and falling near the middle of the second hearth — should begin to burn as it approaches the circumference of the latter. The third hearth should be covered with the characteristic blue flame of burning sulphur, and lively sparking should accompany the stirring. On the fourth hearth the sparking should nearly cease, although the

orange glow will show the increase in heat which accompanies the rapid oxidation of the ferrous sulphide.

This condition still continues on the fifth hearth, though before the ore reaches the central drop-hole its combustion diminishes perceptibly. There is still considerable oxidation and heat on the sixth hearth, though the ore now cools perceptibly as it reaches its circumferential drop-holes and falls into the hopper below.

It is the business of the furnace-man to maintain this typical condition on each of his six hearths and, assuming that the character and feed of the ore, the speed of the stirrers, and the strength of the draught at the uptake flue are suitable and uniform, his principal agents for influencing the behavior of the different hearths are (*a*) regulating the draught from hearth to hearth by removing obstructions from the draft-holes, and opening or closing the doors for the admission of air at the lowest hearth; (*b*) adding carbonaceous fuel to one or more of the hearths so as to correct rapidly any unduly low temperature.

The first of these remedies is much the more common and, indeed, is applied systematically as part of the operation of roasting. It consists in cleaning out at fixed intervals the crusts which form in certain of the drop-holes, and which decrease their size to such a degree as to interfere seriously with the draught. Crusts are formed rapidly on the roof of the third hearth from the sparks which fly up and stick to the brick arch. On the second and fourth hearths the crusting is confined more to the portions of the roof immediately over the drop-holes, and the openings themselves tend to become choked. These accretions have to be barred off every 24 hours, using long chisel-pointed bars, suitably curved. The lumps are pulled out of the doors, or pushed through the drop-holes onto the next floor below to make their withdrawal more convenient. Below the fourth floor but little crusting occurs.

Assuming the furnaces to be in normal condition, the rapidity and completeness of the oxidation depends greatly upon the draught. At the Washoe smelter the measurement of the draught is made in the large flue which belongs to each section of eight furnaces. It varies from 0.5 to 0.6 inches of water, the latter being preferable.

If the temperature of the furnace has fallen below normal and the third hearth fails to show its uniform blue flame of burning sulphur, it is found advantageous to use a little carbonaceous fuel

rather than to suffer from the inconvenience of a considerable amount of poorly roasted product before the oxidation of the ore itself restores the normal temperature. This is usually effected by throwing in some light wood or shavings upon the third hearth, which soon raises the temperature so that the charge begins burning upon the second hearth in the regular manner.

The hoppers which receive the roasted ore discharge their contents into five-ton cars which are partially covered, and are shaped in such a manner as to diminish the escape of dust. These cars run directly over the hoppers of the reverberatory smelting furnaces, into which they dump the hot calcines, no fluxing material being required.

Samples for the determination of the sulphur, and other prominent constituents, are taken with a shovel from the sixth hearth four times on each shift, and are deposited in locked receptacles. They are so arranged that the work is known of each furnace on each shift.

As the McDougall furnace derives its heat entirely from the combustion of the sulphides which it is roasting, a new cold furnace must, of course, be heated by carbonaceous fuel to such a point that the sulphides will ignite, and then maintain the desired temperature. For this purpose, the older furnaces were provided with a small side fire-box at the bottom, arranged for burning coal, and two to three days' time was required to bring the brickwork to a suitable temperature. Later it became the practice to burn the coal directly upon one of the hearths of the furnace itself — usually upon the third hearth; this resulted in a marked saving of both time and fuel. Still more recently it has been found that even better results could be obtained by the use of petroleum products, and that a cold furnace could be brought into working condition in 24 hours, and with a considerable saving in fuel and labor.[1]

[1] Through the kindness of Mr. Wraith, superintendent, I was given the opportunity to witness the starting of a cold furnace with the use of oil as the chief heating agent, and made the following notes: Washoe smelter, Aug. 24, 1909. The McDougall roaster on which the test was made was practically cold, having been stopped for repair a week before. Its hearths were still filled with partly-roasted ore. The feed had been run long enough in advance to bring fresh sulphides down fully upon the second hearth, considerably upon the third, and scantily upon the fourth. Seventeen hundred and thirty pounds (785 kg.) of ordinary semi-bituminous coal had been thrown

ROASTING SULPHIDE ORES IN HEAPS AND STALLS

Since the original publication of the book which the present volume replaces, this branch of metallurgy — then so important — has fallen to such a subordinate position that I do not feel justified in occupying any considerable space with its study.[1]

While recognizing the inconsistency of going through with a tedious, wasteful, and costly operation, for the purpose of throwing away the natural fuel that the ore contained and then purchasing coke to replace it, we had not learned how to oxidize the sulphur and iron of our ores in the smelting furnace in such a manner as to delevop the necessary temperature, and to apply this temperature to the fusion of the ores themselves.

So long as we attempted to "roast" our ores in the same furnace in which they were to be smelted, we made but slight advances in the desired direction. It was not until we learned that the true system was to liquate the sulphides from the gangue, bessemerize the former as they showered down toward the blast, and set the resulting FeO to work eating away the column of silicious gangue, that we began to appreciate the true value of the sulphides in

into the working-doors of the third hearth, and the stirrers had been run long enough to spread it.

At 9.13 A.M. a small wood fire was kindled upon the third hearth to create a local heat sufficient to burn the oil properly.

At 9.35 the oil-feed was started upon the fourth hearth through one of the working-doors, using compressed air at 16 lb. per sq. in. In default of long-flamed crude oil, the less suitable fuel oil was employed.

At 9.45 a long flame was coming up through the central drop-hole between the third and fourth hearths, and the coal upon the third hearth was burning vigorously.

At 10.20 the sulphur was burning freely on No. 2 hearth, and rakes and feed were started, feeding at the rate of 38 tons (34.5 met. tons) per 24 hours.

At 11.15 the entire surface of No. 2 hearth was covered with the blue sulphur flame, No. 3 being obscured by the coal.

At 11.43 the oil was discontinued, 21 gallons (80 litres) having been used during a period of two hours and eight minutes.

More coal was thrown upon No. 2 hearth from time to time during the next few hours, and in 20 hours from starting, the furnace was running in normal condition, having consumed 21 gallons (80 litres) of fuel oil, and 4230 lb. (1918 kg.) of bituminous coal.

[1] Where exceptional conditions render this subject of particular interest to the metallurgist, he is referred to my old "Modern Copper Smelting," especially to the seventh, and subsequent, editions.

replacing coke; and even after we were accomplishing this result successfully, it took us a long time before we could divest our minds of the idea that we were still, in some way or other, depending mainly upon the feeble process of roasting prior to the actual fusion!

As finely divided ores lie in a compact mass and offer no channel for the draft — slight, but indispensable — which supports combustion, it follows as a matter of course that the roasting of sulphides in heaps or stalls is intended strictly for ores in lump form. A moderate proportion of fines is used as a covering, and sometimes as a foundation, but the process itself is for lump ores, and the metallurgist is indeed fortunate if he finds himself able to work up in the same process even the fines which belong to the lump ores he is roasting.

Since mastering the art of bessemerizing and smelting these ores in the blast-furnace, they are now smelted raw in this country, almost universally, their sulphur and iron content supplementing the coke, and permitting the reduction in quantity of the latter to an extent that effects a very important diminution of costs. This is "partial pyrite smelting" — a process peculiarly American, although spreading rapidly to all civilized mining communities.

Our sole material suitable for heap or stall roasting being thus diverted into another channel, these processes are becoming obsolete. Still, exceptional conditions may demand their employment, and I will point out the more important principles upon which they rest.

Omitting exceptional cases where the ore is so poor in sulphides as to require an admixture of chips, or brush, or coal to enable it to support the temperature essential to the process of oxidation, heap roasting may be regarded as a completely self-sustaining operation. The scanty layer of wood upon which the heap is built corresponds merely to the kindlings that are required to start the combustion of a stoveful of anthracite coal. It takes a certain degree of heat to start the sulphides burning; but, once started, they generate ample heat to keep the process alive. Indeed, if uncontrolled, they would generate heat so rapidly that they would fuse themselves into a solid lump of low-grade matte, intermixed with fragments of the infusible gangue, and their oxidation would cease.

It is plain, therefore, that their combustion requires intelligent

control, and as experience has taught us many lessons in regard to the laying-out of the roast-yard, building and removing the heaps, etc., it will facilitate the study of the subject to adopt a systematic classification of its more important features:

Heap Roasting

(*a*) Selection and preparation of ores
(*b*) Selection and preparation of roast-yard
(*c*) Erection, management, and removal of heaps

Stall Roasting

(*a*) Construction
(*b*) Management of process
(*c*) Comparison with heap roasting.

Heap Roasting. — (*a*) Selection and preparation of ores. As it is essential to the success of the process that the interstices between the fragments of ore in the heap shall remain reasonably open, it is manifestly impossible to roast in heaps ores which will not conform to this requirement. Ores already in a fine state of subdivision (fines) are debarred; consequently, ores which decrepitate by heat, and fly into small particles, are equally unsuitable. Again, there are certain pyrite ores whose physical make-up is of such a nature that they resist the process of gradual oxidation. Their surfaces will ignite from the influence of the kindling wood, but will not support combustion independently, and will soon go out. These ores are not common, but occur with sufficient frequency to make it wise to bear them in mind. Pyrrhotite, so far as my own experience and observation extend, roasts in heaps as well as pyrite, although requiring slightly different management. It also roasts perfectly well in stalls — but not in kilns.

Zinc blende ignites at a tolerably high temperature, but then burns vigorously for a short time, and goes out as soon as a more or less thick layer of oxide and sulphate has formed on the surface of the lump. In connection with plenty of pyrite, and broken to a moderate size, it roasts reasonably thoroughly, the slow combustion of the reliable pyrite furnishing the heat required to keep up the reluctant oxidation of the interior of the blende.

Galena melts easily, roasts but little, and, in the small proportion encountered by the copper smelter, receives no attention.

Chalcopyrite is quite fusible and, if present in large quantity, would demand unusual care; but, in the proportions usually en-

countered, it roasts well, its own deficiencies — as in the case of most of the other sulphides — being helped out by the admirable behavior of the pyrite.

The behavior of the richer ores of copper is unimportant. In moderate proportion, with ample pyrite, they roast sufficiently well, although they usually carry so much copper that this is really a matter of indifference. If they are present in very large quantities, they would not require roasting at all.

The easiest and safest way to determine whether a new ore is suited to the process is to erect a heap with all the precautions that would be observed in regular practice, and using a very small amount of kindling wood, unless the proportion of sulphides is small. If the heap melts into a block of matte during the first 48 hours, as is likely to be the case under inexperienced management, the result should be repeated with half the quantity of wood, and with still more careful regulation of the draft. If it is kept burning slowly, and ample time is allowed it to work out its own results, an excellent idea can be obtained of the behavior of the ore under examination.

Such a test will also give a good idea of the rate of roasting, and of the maximum size of the lumps which may be permitted.

No rule can be laid down in regard to this latter point. Some ores support combustion so freely that they will burn to the center in lumps the size of one's head, while others must be reduced to egg-size if a thorough roast is desired. In the vast majority of cases, it may be said that the ore should be broken to pass a hole of about 2½ in. diameter (0.064 m.), although, of course, most of it will be decidedly smaller, and much of it will pass a one-inch (0.025 m.) opening, and thence graduate down to absolute dust.

For reasons connected with the process itself, it is found that more economical results can be obtained by graduating the sulphide ore into at least three sizes. Assuming the product of the original mine-crusher to be of a size that will all pass a 2½ in. ring, the next-sized screen may have holes one inch in diameter, while the fine screen should have openings with a diameter of about one-fourth inch. The three classes of ore which thus result are termed, respectively, coarse, ragging, and fines.

The suitable classifying of the ore in this manner is one of the most important steps in the roasting of ore in heaps or stalls.

An ordinary cylindrical punched screen of boiler-plate, making

12 revolutions per minute, and with a maximum fall of one inch to the foot (one in twelve), will separate easily 20 tons per hour.

(*b*) Selection and preparation of roast-yard. The main points to bear in mind in laying out a new roast-yard are: situation in relation to the mine, so that the ore may not have to run up hill to reach the elevated tracks above the roast heaps; situation in relation to the smelter, so that the ore may not have to be elevated to the smelter bins; safety from floods, or violent winds; situation in relation to the town, or other points, so that the gaseous products may be as little offensive as possible. The first two of these conditions are not always attainable, and careful consideration must then be given to the cheap transportation of the ores to and from the heaps.

The accompanying illustration gives a general idea of a suitable roast-yard, the elevated track which delivers the ore crossing all the heaps at right angles to their longitudinal axes, while a lower track running along one side of the yard makes it convenient to load cars with the roasted ore.

If the ground permits, the most convenient form for the yard is that of a long, narrow rectangle, the width of the latter being determined by the length of the heaps, while its length corresponds to the number of heaps for which it is intended.

The length that this yard must have to roast a given number of tons of ore per 24 hours may be determined at once, providing one has learned, by experiment, the size of heap suited to the ore, and the length of time required for the complete roasting of the heap. For instance, assume that we have decided that a suitable size for the heaps would be 24 ft. in width, 140 ft. in length, and 6 ft. in height,[1] and that such a heap would contain 240 tons of raw ore, and require 70 days for complete roasting. Adding to this 10 days for removal and rebuilding (the blast-furnaces being supplied from several heaps simultaneously), we have 80 days for the completed cycle. Therefore, as one heap roasts 240 tons in 80 days, it will roast $\frac{240}{80} = 3$ tons daily; and if we desired to roast 100 tons daily, we should require $\frac{100}{3} = 33\frac{1}{3}$ — say 35 — heaps.

As the heaps are to be 40 ft. long by 24 ft. wide, it will be suitable to increase this area to 60 × 36 ft., to afford the necessary space for working. This means, therefore, that the yard must be 60 ft. wide and 36 × 35 = 1260 ft. long. The total area of the

[1] One foot = 0.3048 m.

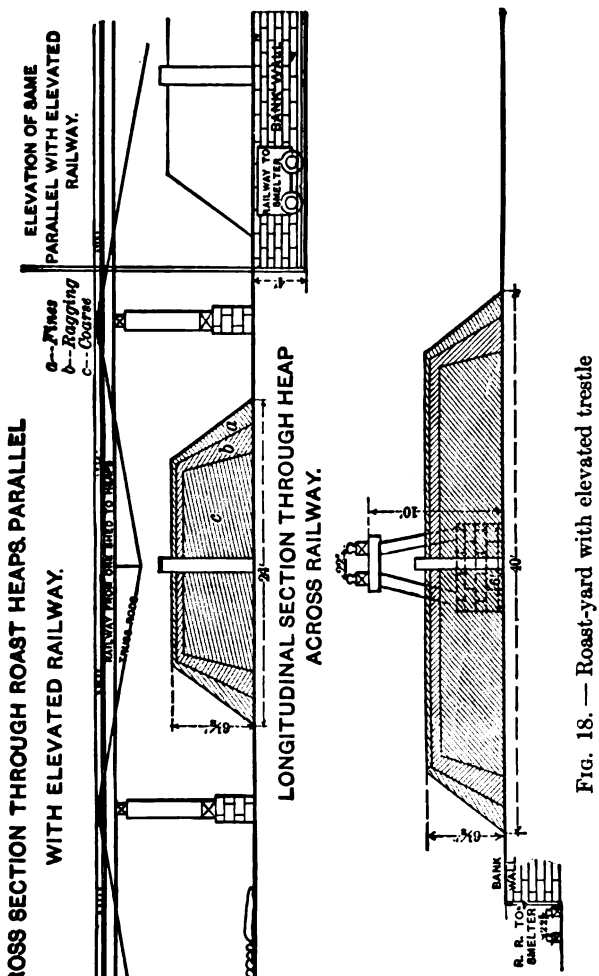


FIG. 18. — Roast-yard with elevated trestle

rectangle will be 1260 × 60 = 75,600 sq. ft. As it takes 75,600 sq. ft. of ground to roast 100 tons of our ore per 24 hours, it will take 756 sq. ft. of yard for each ton of ore roasted per 24 hours.

Thorough drainage and protection against floods are the most important features in the preparation of the yard. The surface soil should be removed down to gravel or hard-pan, and be replaced by a layer of concentrator tailings, sand, or granulated or crushed slag; the latter material being the best, as it will not disturb the blast-furnaces if accidentally shoveled up with the roasted ore. The areas to be occupied by the heaps may be covered several inches deep with sulphide fines, which will gradually become oxidized, and may then accompany the roasted ore to the smelting furnaces.

(*c*) Erection, management, and removal of heaps. The building of the heap itself should always be conducted by a man experienced in this work. The amount of cord wood used should be the very smallest quantity that will give the roast a good, vigorous start. Anything beyond this is not only a waste, but is harmful to the process. The ore arrives in cars on the elevated track, whence it is dumped in the free space between the heaps, or upon planks placed temporarily upon the heap. It will not answer to dump it direct upon the heap itself, as the constant impact consolidates it too much, and interferes seriously with the quiet uniform combustion which is the essential feature of the process.

After the proper quantity of coarse ore has been formed into shape, a layer of ragging is placed over it — thick at the bottom, and thinner at the top, as is shown in Fig. 18. The proper quantity of fines is then accumulated in the free space between the heaps, and the wood is fired. As the burning progresses, the heap is gradually covered with fines, much care and experience being needed to keep the combustion slow, without discouraging it completely. This is accomplished mainly by regulating the admission of air through the layer of fines near the base of the heap and, especially during the first few days, requires constant intelligent supervision. The details of the burning are studied at length in the work already cited.

As the heap undergoes oxidation, it swells and increases in size, constantly fissuring its enveloping layer of fines, which are as constantly mended and renewed. It should be allowed to cool for

several days after it has burned itself out, as it cannot be torn down and loaded for the smelter with either comfort or economy while it is still very hot. The unoxidized outside layer of fines is usually stripped off and used for covering the next heap. Although a certain proportion of the fines, during their employment for covering and for foundation, gradually become sufficiently roasted to be sent to the smelter along with the coarser ore, a much larger part remains unoxidized and causes a progressive accumulation which is always one of the most embarrassing features of this process at plants where there are no roasting furnaces available. This subject is of such importance that I devote an especial chapter to its study.

The results of heap roasting are peculiarly dependent upon the care and skill with which the process is conducted. With reasonably favorable ores, one may expect to roast 92 per cent of the original weight of ore in the heap, and to be left with 8 per cent of fines which have lost but a small proportion of their sulphur. If the ore contained 32 per cent sulphur originally, the resulting roasted material might contain 7 per cent of sulphur. These figures represent an approximate average of large quantities of different ores, all of which were roasted with care, and stripped thoroughly of their unroasted covering. In many of the samples, the condition of the copper at the termination of the roast was determined, with the following approximate average results:

8.2 per cent of the copper was in the form of sulphate.
11.4 per cent of the copper was in the form of oxide.
80.4 per cent of the copper was in the form of sulphide (by difference).

If the stripped heaps are exposed to the wet, the loss of copper is certain to be large. In addition to the soluble copper sulphate formed during the operation of roasting, this salt is formed with great rapidity in the roasted ore under the influence of moisture, and would ordinarily prohibit the employment of the process in wet countries, especially if the climate is warm as well. Wendt gives some important figures relating to the losses in heap roasting [1] as practised at Ducktown, Tennessee, where the rainfall is heavy. Owing to the unexplained heavy losses in copper during treatment at the Ducktown smelter, Wendt undertook extensive

[1] "Pyrites Deposits of the Alleghanies," by A. F. Wendt, New York, 1866.

experiments to determine their origin, and finally located much of the loss in the roast-yard, where the copper was leached out as sulphate. I append the result of one of his determinations:

PILE NO. 447. — MARY ORE

Gross Weight of Ore	Per Cent Water	Per Cent Copper	Fine Copper, Pounds
172,882	3.0	4.7	7,881
1,532	5.5	6.3	91
198,800	2.0	4.5	8,767
32,178	4.0	5.3	1,637
26,865	5.5	4.6	1,167
32,245	3.0	6.2	1,939

464,502 gross pounds ore contained 21,482 pounds copper.

"Weight of the roasted ore (wet) was 495,566 pounds, assaying 2.85 per cent, or 14,126 pounds fine copper. During an exposure of 186 days the ore had lost 34.3 per cent of its copper."

As it is usually impracticable to protect a large roast-yard from the weather, it is customary to turn to stall roasting in wet climates.

The operation of heap roasting lends itself fairly well to contract work, the contractor receiving the raw ore in cars on the track above the heaps, roasting it, and delivering it in cars ready to be taken to the smelter bins. His contract should include premiums for extra well-roasted ore, and heavy deductions for the delivery of any that is poorly roasted. As it is impossible to take fair samples of the roasted ore without too much trouble and expense, the degree of the roasting is judged by the grade of the matte produced from the blast-furnace.

With the high wages of Western smelters, the cost of heap roasting per ton (0.907 m.t.), including the loading of the finished product, may be from 55 to 60 cents. In the severe climate of Sudbury, Ontario, with low wages, I paid 22 cents per ton for unloading the raw ore, building and covering the heaps, and attending them until the completion of the burning. For stripping, tearing down, and loading the roasted ore on cars, and unloading the cars by hand into the smelter bins, I paid 16 cents per ton of roasted ore. The company furnished wood, tools, and barrows, but the contractor sharpened his own picks and bars. On the above basis the contractor made a reasonable profit. The only laborers he could depend upon to work through the stormy nights of this northern region were Finns, Poles, and Russians.

Assuming the ore to be reasonably well suited to the process of heap roasting, the results obtained depend mainly upon the roaster foreman.

The heap roasting of matte is too imperfect, wasteful, and expensive an operation to demand consideration. If information upon the subject is required, it may be found in old "Modern Copper Smelting."

Stall Roasting. — (*a*) Construction of stalls. The employment of this process is, of course, limited by the same considerations that apply to the preceding subject, and detailed information regarding it must be sought in older metallurgical works.

Stall roasting may be regarded simply as an improved modification of heap roasting, in which the heap is enclosed by permanent walls, kept in a circumscribed area, provided with a more efficient draft, protected from the weather if desired, and placed upon a generally more civilized basis.

For economy of space, material, and heat, stalls are usually constructed side by side, in a long row — or rather, in two rows, back to back, with a central flue running longitudinally to a stack of moderate height. Each stall is connected with the flue by a number of draft-holes in its rear wall.

As it is impracticable to secure uniform oxidation of the ore in large stalls, it is customary to build them of small size; and this is done the more willingly, as the roasting operation is thereby completed in a few days.

A reasonable size might be 8 ft. long, 6 ft. 6 in. wide, and 6 ft. high. Such a stall will contain 20 tons (18.14 m.t.) of heavy sulphide ore, and will burn for about 10 days. The accompanying illustrations show the stalls formerly in use at the Parrot smelter in Butte, Montana.

They are built of slag brick, and the ore is brought to them on a small track running along above the main flue, and serving each individual stall by means of temporary turn-sheets and short tracks. The main flue is two feet wide in the clear,[1] and connects with a stack 82 ft. high and 3 ft. 6 in. square inside. This stack furnishes a sufficient draft for 50 stalls, arranged in a double row. For the better admission of air to the ore, there are a number of

[1] One foot = 0.3048 m.

small openings in each side-wall, which connect with an open channel in the division-wall. The stalls are not ironed, and need much care and repairs to maintain them in suitable condition. A track to the smelter runs along the front of each row.

(*b*) Management of process. The secret of successful stall roasting is to use a minimum amount of wood to kindle the ore.

On the level floor of the stall, three or four channels are formed with lumps of ore and filled with kindlings. The very small bed of wood is then laid down, the stall filled with ore (the front being closed with a heavy sheet of iron), and a layer of ragging and fines

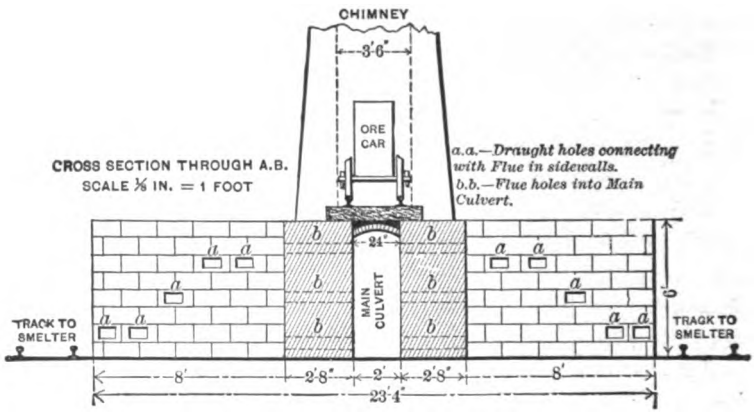


FIG. 19. — Roast stalls for ore — section

placed on top, the desire being to encourage the draft through the stack, and to prevent any objectionable fumes from the stall itself.

The management of the operation is based upon the same general principles as heap roasting, the object being to secure a uniform and thorough combustion at a comparatively low temperature.

The results and cost of the process are practically the same as for heap roasting. The cost of this battery of 50 stalls, with stack, at Butte in 1883, was about $3300. The slag brick of which it was constructed were made on the spot, at the company's reverberatory furnaces, and thus furnished an unusually cheap material.

The stall roasting of matte is peculiarly unsatisfactory and costly, this substance requiring repeated burnings to remove any reasonable proportion of its sulphur. We have, at the present

time, methods which are so much more efficient and economical that I can scarcely imagine conditions under which it would be advisable to resort to it.

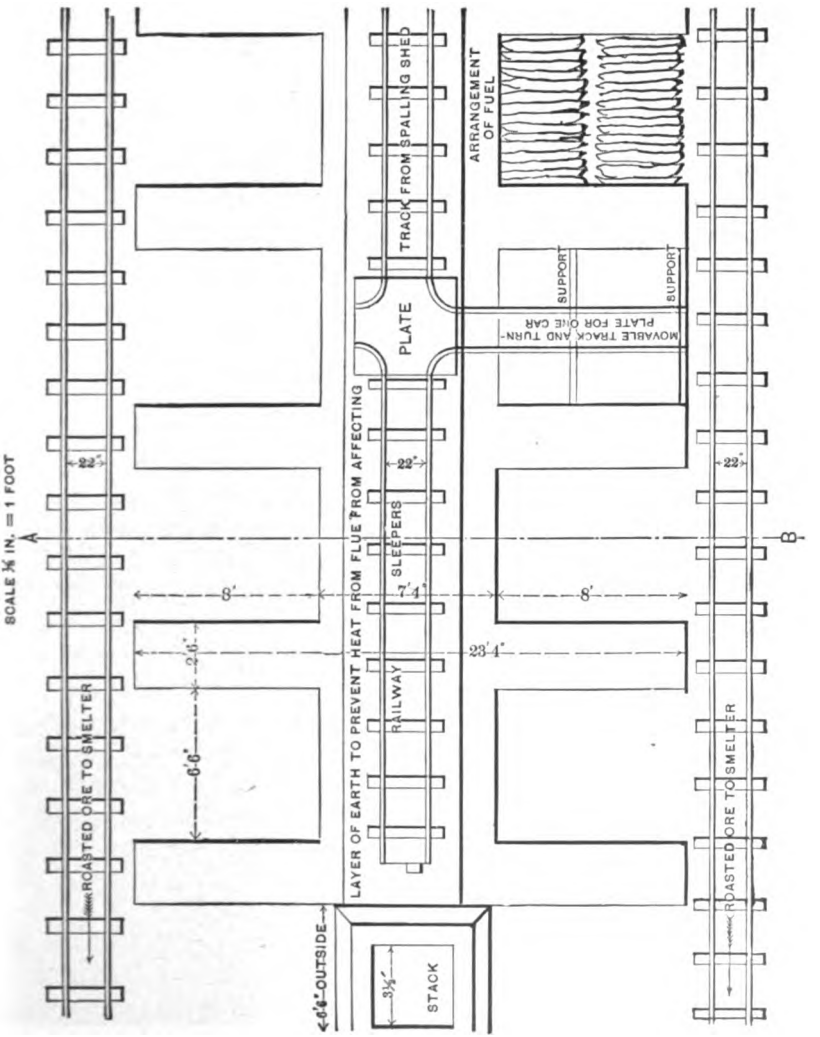


FIG. 20. — Roast stalls for ore — plan

(*c*) Comparison of heap and stall roasting. While the cost of these two kindred operations is pretty nearly the same, the stall offers the greater advantage in all but exceptional cases. Its

two strongest claims are its small waste of values, and its rapidity. The whole process is compressed within a very small, paved area, and may be roofed over without undue expense. This alone would establish its advantage in a wet or windy climate. Being made up of a number of small units, the operation is completed within two weeks, instead of taking from two to nine months. This means an immense saving in the amount of capital tied up in ore.

Even the first cost of construction is seldom much in favor of the heap, as the expense of preparing the large roast-yard, with its great length of tracks and trestles, is likely to be full as much as the building of the compact block of stalls.

The conditions which would speak most strongly for the employment of heap roasting are a dry, equable climate, level gravelly soil, and absence of skilled labor.

CHAPTER VI

THE BLAST-FURNACE

THE blast-furnace is one of the oldest types of metallurgical apparatus, and was in use before the beginning of written history — although, naturally, in an exceedingly rude form.

In this furnace the ore and fuel are in contact, and combustion is intensified by the use of a forced blast of air. It is suited mainly for the employment of concentrated, carbonized fuels, such as coke or charcoal, the use of the latter being comparatively rare in copper smelting, owing to its cost, fragility, and inefficiency.

In order to study the processes of blast-furnace smelting clearly and concisely, it is necessary to separate them into two classes, the division being based upon the character of the process itself and not upon any peculiarity in the form or construction of the furnace in which the process is conducted.

These two distinct processes are:

(*a*) Smelting the ore with carbonaceous fuel.

(*b*) Smelting the ore (mainly) with the heat derived from the oxidation of its own sulphur and iron. (See Chapter VIII.)

(*a*) *Smelting the ore with carbonaceous fuel.* — This operation is conducted in a reducing atmosphere so far as the constituents of the ore are concerned; for, although nearly sufficient air is blown in to burn the coke, there is no excess of oxygen; and, as the carbon has a powerful affinity for oxygen, it consumes the entire oxygen-content of the blast, and even then craves still more, as much of the carbon can only secure oxygen enough to burn to CO. Consequently, the furnace-shaft is filled constantly with heated CO gas which, being still desirous of taking up an additional atom of oxygen and burning to CO_2, acts as a strong reducing agent upon such metallic oxides as the ore may contain. In this way copper oxides are reduced and saved from being slagged by the silica, while the higher iron oxides are reduced to FeO, and thus fitted to combine with the silica and form slag.

This operation is, therefore, essentially a process of reduction,

and is not suited to the fusion of raw sulphide ores, as their sulphur — not being able to obtain oxygen and thus burn to SO_2 — would continue, for the most part, united with its original metals, and would thus cause the production of an excessive amount of matte, low in copper.

A charge composed chiefly of oxidized ores is almost characteristic of this process of blast-furnace smelting with carbonaceous fuel.[1]

(*b*) *Smelting the ore with the heat derived mainly from the rapid oxidation (combustion) of its own sulphides.* — This is a process of recent date, and accomplishes its results in a manner so radically different from the older method that it would only obscure matters to attempt to study these two totally diverse subjects at one and the same time. This new method is called "pyrite smelting,"[2] and, from its very nature, calls for an oxidizing atmosphere that shall be sufficiently thorough and extensive, not only to burn the small amount of coke which may be used, but also to oxidize vehemently the required proportion of sulphides contained in the ore-mixture. It is a subject of great importance, and will demand a separate chapter.

The present chapter, therefore, deals solely with ordinary blast-furnace smelting with carbonaceous fuel and a reducing atmosphere, and will include the construction of the furnace itself.

While the copper blast-furnace is a familiar object to most of the readers of this book, a brief recapitulation of the fundamental principles upon which it operates will assist in clearing away the complications which surround the modern furnace, and which are due to improvement and amplification of detail, and not to any addition or modification of the general principles of the process.

The essential features of all copper blast-furnaces — whether ancient or modern — are: a more or less vertical shaft, in which the fuel and ores are placed in alternate layers; a blast system, by which moderately compressed air is forced through openings

[1] The manner in which these ores may have become oxidized is immaterial. The oxidation may result from the slow processes of nature — as in the upper zones of sulphide deposits; or it may be the rapid work of man — as in the roasting of sulphide ores.

[2] The term "pyritic smelting" was, long ago, applied by Dr. Percy to a totally different process in which there was no attempt at oxidation of the sulphides.

(tuyere-holes) near the lower extremity of the shaft; a device for the separation and removal of the melted products as they flow out near the bottom of the furnace; and, subordinately, a system for the removal of smoke and fumes, often combined with means for rescuing the fine particles of ore (flue-dust) which are carried out by the air current. These originally simple, and comparatively cheap, requirements have been amplified and developed, of late years, in an astonishing manner, and the main purpose of this chapter is to point out the methods of construction and management which have placed the blast-furnace of the copper smelter upon the same plane as that of the iron master, and have reduced the cost of treatment in a corresponding degree.

The most important improvements have taken place during the past 35 years, and consist in the general substitution of metal water-jackets for the destructible fire-brick walls of the older furnaces; and the almost complete abolition of hearth-accretions and freeze-ups by the substitution of the external forehearth, or settler, for the interior crucible of former days. These are the two fundamental inventions which have emancipated the copper furnace from its earlier restricted condition, and which have rendered practicable the application of two further improvements, without which the present development of smelting would have been impossible. These are: first, the recognition of the fact that *mass* is one of the controlling factors in smelting operations, and that a large furnace is easier and cheaper to run than a small one; second, the application of modern methods of handling material, without which the employment of these large furnaces would be impossible.

According to almost universal practice at the present day, the ore is smelted with the necessary coke in a water-jacketed furnace of considerable size, while the melted products — without any attempt at interior separation — flow continuously into an exterior, independent forehearth or settler. This settler is expected to be of such a size that the superheated stream of melted products from the furnace shall keep it liquid and open, without being so hot that its walls are endangered, while it serves also as a matte-reservoir, or regulator, between the blast-furnace and the converters.

In the slow, unruffled passage through this settler, the matte globules find an excellent opportunity to separate thoroughly

from the slag and accumulate, while the worthless slag flows in a continuous stream over the spout at the top; the matte is, of course, tapped from the bottom of the forehearth as needed.

The copper blast-furnace has attained its present convenient form by the united efforts of a great number of practical metallurgists and machinists. The great smelting-machinery plants of the world now turn out these furnaces complete, much in the same manner in which they would furnish steam engines. Various types have become standardized and, until the metallurgist sees — from the results of many-sided experience — that he can modify existing furnaces to decided advantage, it is wiser to select some standard type and not complicate a new enterprise with the serious difficulties and drawbacks of experimental work. He is bound to have several unknown quantities to deal with, at the best, and he is unspeakably foolish to add to them unnecessarily.

The simplest type of water-jacket is the original circular form, consisting of a double-walled shell of steel plate, or wrought iron, and appearing like a tapering boiler set on end. The accompanying illustration, Fig. 21, shows the general arrangement of a little furnace of this simple description with self-feeding water tank, as arranged by Bartlett in the seventies.

The difficulty and expense of running such small furnaces as this have caused their complete disappearance under ordinary conditions; but such a little plant would answer for experimental purposes, or for temporary service at a small, isolated mine.

As may be seen, the furnace is self-contained and self-supporting. The water-cooled shell extends in one piece from just below the wind-box to the level of the feed floor, the total height being from six to nine feet (1.83 to 2.7 m.). The diameter of the shell increases toward the top at the rate of about one inch per foot of vertical length (1 in 12). The tuyere openings consist of cast rings inserted into the narrow water-space, a circle of rivets holding the inner and outer shells in close contact with these castings. Hand-holes, for cleaning the water-space, are arranged in a similar manner in the outer sheet. The water-space at top and bottom, as well as the slag-opening, may be closed by wrought-iron rings, to which the two shells are riveted; or the inner shell may be flanged over and riveted to the outer, forming a right angle to the vertical axis of the furnace. The

bottom may consist of drop-doors like a foundry cupola, or of a simple cast-iron plate, dished to prevent cracking, and bolted to the bottom ring of the shell; or, where an inside crucible is desired, it may be built up from the ground in the shape of a brick hearth

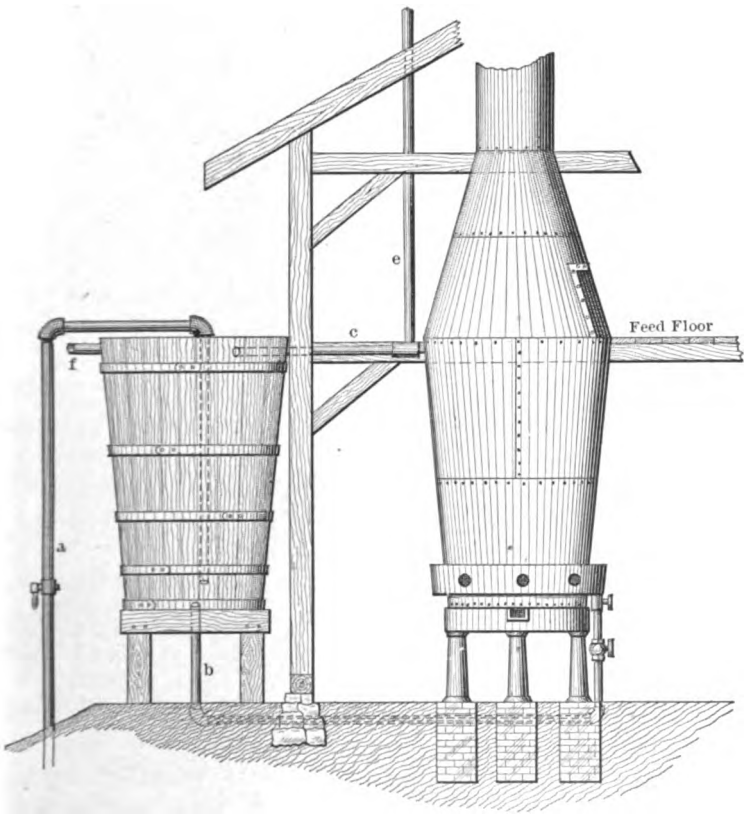


FIG. 21. — Early type of water-jacket furnace

of circular, oval, or rectangular form, held together by an iron shell. Furnaces of the above type, with cast-iron bottom plate, may thus be constructed without using a single fire-brick, as clay is, under certain conditions, a sufficient protection for the bottom, as well as for the hood which connects the shell with the sheet-iron chimney.

While no one would think of employing such an apparatus as this for any permanent work, its low first cost, rapidity of trans-

portation and erection, independence of fire-brick, and general flexibility may render it invaluable in a remote district where small quantities of comparatively rich ores must be beneficiated on the spot to raise money for more extensive operations.

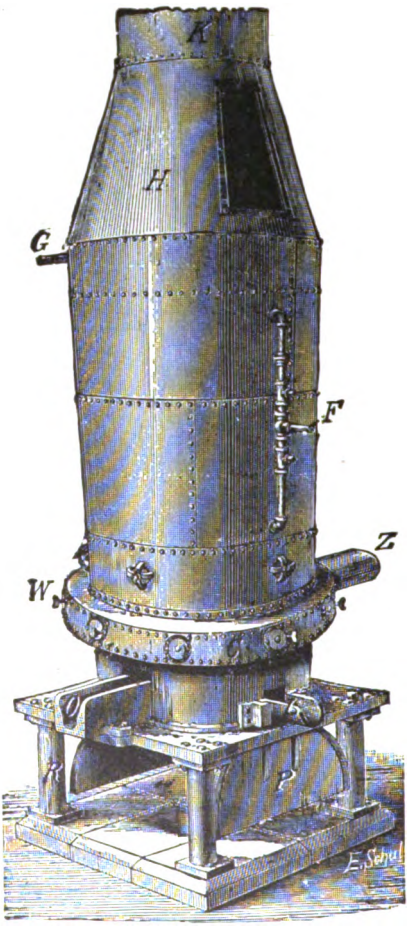


FIG. 22. — Furnace for oxidized ores — elevation

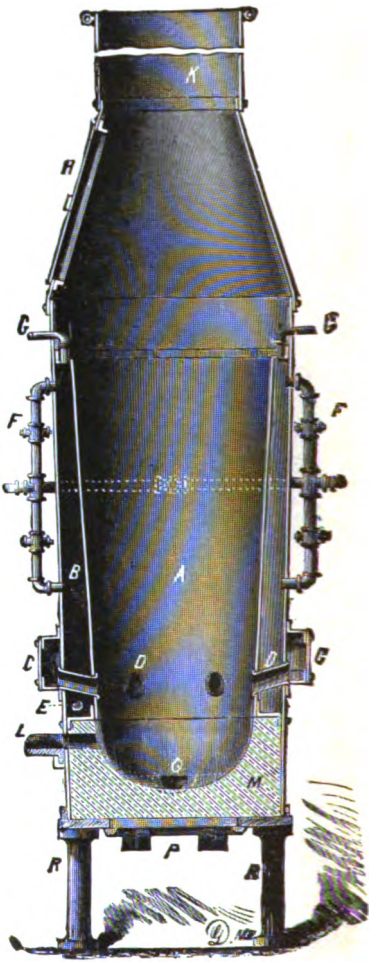


FIG. 23. — Furnace for oxidized ores — vertical section

An interior crucible is employed in cases where the melted products seem too cold and sluggish for outside separation. This condition may arise either from the fact that the small furnace

does not smelt rapidly enought to deliver a sufficient stream of superheated slag and matte to keep the forehearth warm, or it may come from the nature of the molten products. Slags which are at all refractory need to flow in a stream of considerable volume; and still oftener, rich matte, or metallic copper from oxidized ores, chills quickly, and needs the protection of the interior crucible.

Figs. 22, 23, 24, 25 show the type of small, round water-jacket furnace with interior crucible which smelted into metallic copper a large amount of the carbonate ores of Arizona not many years ago, and which is still used in new regions.

As the penetration of the light blast used with these small furnaces is seldom more than 24 inches (0.61 m.), it is evident

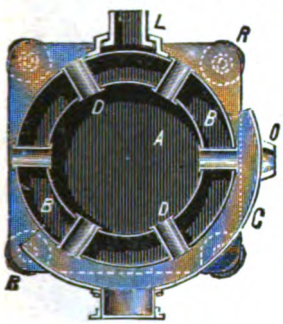


FIG. 24. — Furnace for oxidized ores —cross-section

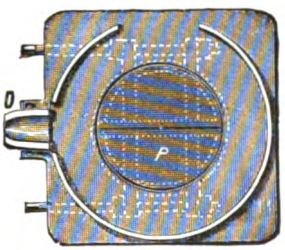


FIG. 25. — Furnace for oxidized ores — cross-section

that the diameter of the shaft, at the tuyeres, cannot much exceed 48 inches (1.22 m.). The most convenient way to increase the area of the furnace, while still retaining the shell in a single piece, is to construct it in a more or less oval form, which gives it sufficient strength to withstand the severe strains to which it is subjected.

The accompanying cuts illustrate a particularly well-designed furnace of the oval type, called the Herreshoff furnace. It is also provided with a convenient movable, water-jacketed forehearth which, when in position against the breast of the furnace, makes a tight, water-cooled passage for the slag and matte to enter the settler, and which is so arranged as to trap the blast, and thus prevent its escape through the breast. It is, of course, not intended for slow-running, refractory slags, nor for mixtures yielding metallic copper, or small quantities of high-grade matte.

The limit of the self-contained, single-piece furnace is soon reached, and, where greater capacity is required, we increase the length of the shaft by placing a number of water-jackets side

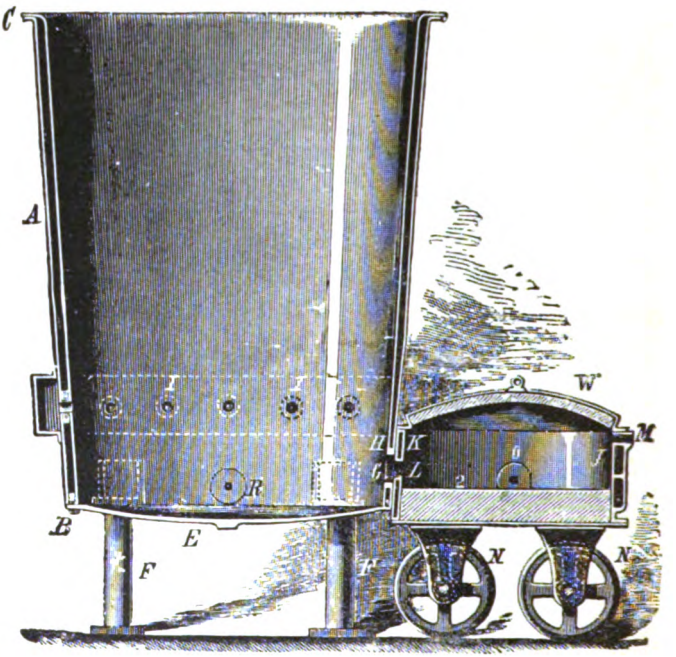


FIG. 26. — Furnace with trapped forehearth — vertical section

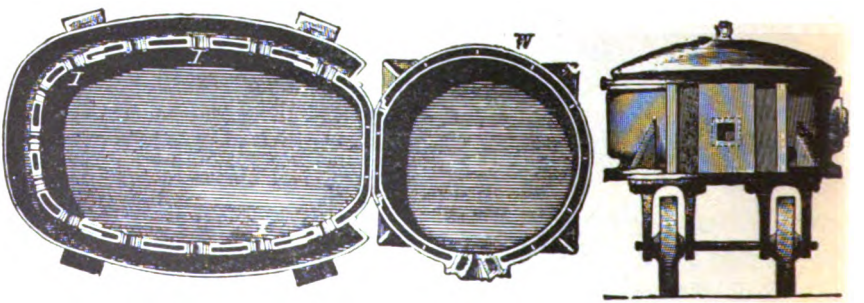


FIG. 27. — Furnace with trapped forehearth — cross-section

by side in sections, or panels. Experience has taught that the rectangular form is the most suitable and convenient, and nearly all modern furnaces having a capacity of more than 100 or 150

tons (91 to 136 met. tons) per 24 hours are constructed in this manner.

The inside width, as already explained, is limited by the power of penetration possessed by the blast. This factor is modified by the condition of the charge — whether coarse or fine — and by the power of the blowing apparatus. The width of furnaces, so far as I know, varies between 33 in. and 56 in. (0.84 to 1.4 m.), although even these extremes may be passed in exceptional cases.[1]

The length of the furnace, being subject to no such limitations, has increased from year to year, always accompanied (under suitable conditions) with a still greater proportional increase in capacity, ease of management, and diminution of costs. In 1905 Mathewson built two furnaces 51 ft. (15.54 m.) in length at the Washoe smelter, which proved so advantageous that a third one 87 ft. (26.5 m.) long was soon added, and is now being increased to a considerable extent. These very large furnaces demand, of course, a corresponding equipment. The buildings, the blast, the ore-storage, the charging system, the settlers, the devices for removing slag and matte, and — above all — the ore-supply must be in harmony with the capacity and requirements of the furnace itself. Such conditions are uncommon; but, when they can be established, it appears that the advantageous limit of length has certainly not yet been exceeded, and has probably not been reached.[2]

I think, then, that we may assume that the length of the copper blast-furnace need be limited solely by the factors already enumerated, and that the elimination of most of the end-jackets, the flexibility of action, and the general ease of management and repairs all favor the use of a single long furnace in place of several of smaller size.

Recognizing, however, that the conditions warranting its use are exceptional, and that most plants must content themselves with furnaces which — although actually large — are small in comparison with those just mentioned, I shall devote the greater

[1] See table of blast-furnace dimensions on pages 146, 147.

[2] The strongest testimony to the efficiency — or non-efficiency — of a furnace often comes through the furnace-men themselves, whose judgment is more apt to be unbiased than that of the inventor or his assistants. I have found the Washoe foremen and furnace-men unanimous in their opinion of the advantages possessed by the large furnaces over the 15-ft. (4.57 m.) ones which they replaced.

part of my consideration to the medium-sized furnace, which is likely to remain the commoner type. Indeed, the very large furnaces present few novel features except size, as their construction demands, for the most part, merely a reduplication of parts.

The modern blast-furnace, even with its great complexity of detail, may be dissected into a few simple, fundamental parts

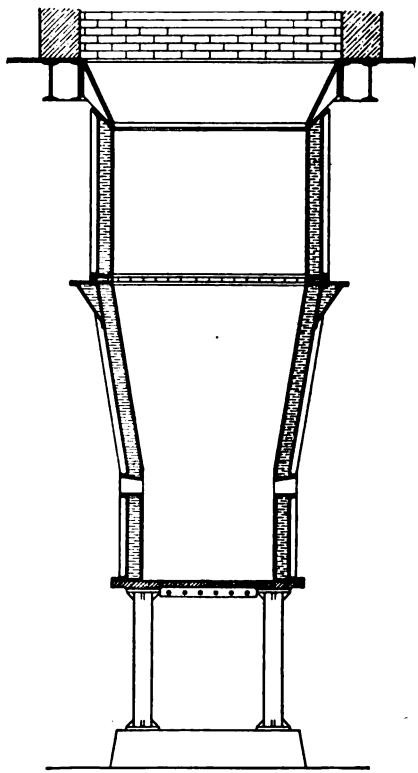


FIG. 28. — Water-jackets — vertical section

which, when once thoroughly understood, may be invested with a multitude of accessories without obscuring the real simplicity of its construction or of its duties.

The most important of these points are:

1. The water-jackets and connections.
2. The hearth, with blast-trapping device.
3. The forehearth, or settler.

1. *The water-jackets and connections.* — The rectangular shaft is formed by the assembling of a sufficient number of separate jackets to enclose a space of the desired size. As the width of the space cannot exceed a given dimension — say, 56 inches (1.4 m.) — the ends may be closed by a single jacket — occasionally two jackets. The long sides, however, are made up of any desired number of jackets placed side by side in panel form, while height may be obtained by erecting a second, or even third, tier of jackets on top of the lower set.

The accompanying cut gives a vertical section of the general arrangement of the water-jackets of a modern copper furnace.

The most important point to notice in this illustration is the complete independence of the system of jackets. It stands alone and unconnected with the brickwork of the charging-space above, and is not required to bear the slightest weight of either brickwork, hood, flue, downtake, stack, or any portion of the upper construction.

The jackets consist of two tiers, the upper tier resting upon the lower (or supported on small adjacent I-beams), and the lower tier supported by I-beams not shown in the illustration.

All of the superstructure rests upon steel girders — shown in section — and the girders are supported by iron columns which, in their turn, rest upon stable foundation. (See the succeeding cut.)

The lower tier of jackets is made with a moderate taper (bosh), beginning shortly above the tuyere openings, and continuing throughout the remaining height of the jacket.

Figs. 29 and 30 show the same set of water-jackets, with the addition of some of the more essential portions of their equipment.

The slightly curved hearth of fire-brick — to protect the bottom-plate — has been constructed, with its emergency tap-hole, to empty the hearth completely in case of accident or repairs. The trap-spout has been bolted in position and, having its overflow-lip considerably higher than the slag-opening which communicates with the hearth, will thus keep this outlet submerged (trapped) in a liquid bath, and prevent any blowing-through of the air blast.

In the illustration, the lower longitudinal tier of jackets consists of three panels, each panel containing three tuyere openings. The upper tier of jackets — being exposed to less severe condi-

tions — are broader, consisting only of two panels, and being supported by lugs which rest on adjacent I-beams and are entirely independent.

The charging-doors are shown at the level of the charging-floor, and the free interval between the upper edge of the superior

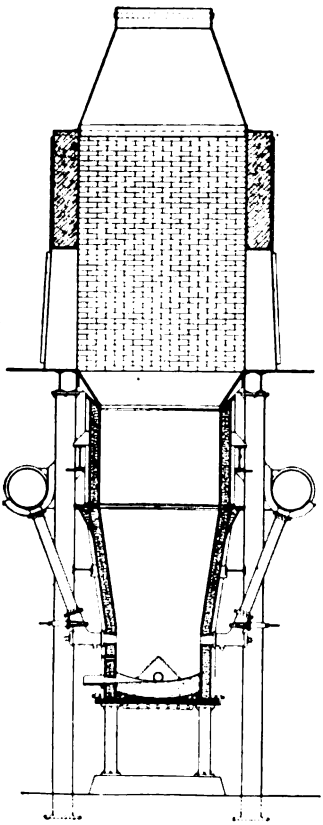


FIG. 29. — Standard matting furnace — vertical section

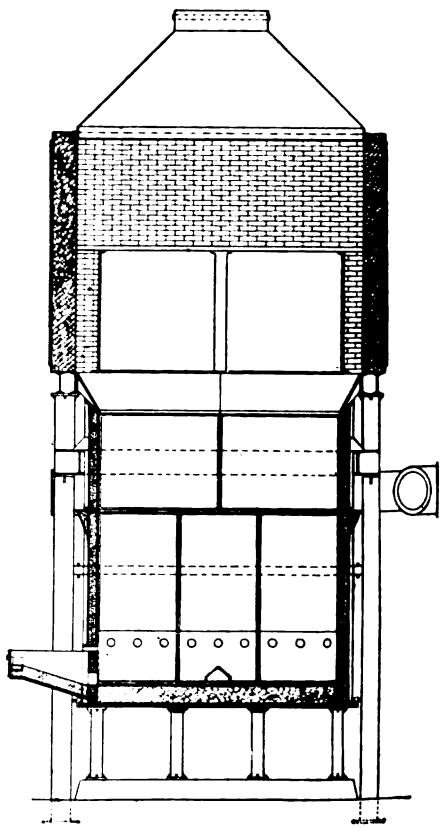


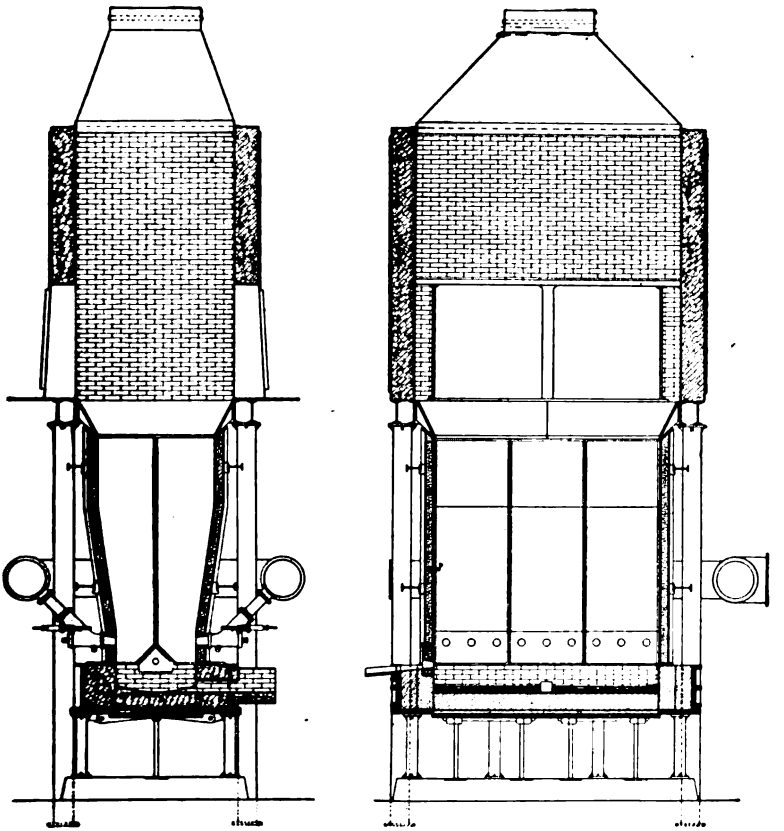
FIG. 30. — Standard matting furnace — vertical section

jacket and the brick casing above is covered by inclined iron plates.

Figs. 31 and 32 illustrate the slight variations which are introduced by constructing the furnace with an inside crucible and — incidentally — closing the ends with two jackets instead of one; also by forming the shaft with one tier of long jackets instead

of two tiers of shorter ones. As in the former illustrations, the bosh begins just above the tuyere openings, but, in order that the shaft may not attain too great a width, it is straightened out again for the upper third of the jacket.

As the object of the inside crucible is to enable us to melt



FIGS. 31-32. — Matting furnace with inside crucible — vertical section

ores which are inclined to chill — such as those which produce metallic copper — the jacket is only continued for a short distance below the tuyeres, where it is replaced by a thick non-radiating brick wall, heavily ironed to withstand the great expansive force exerted by highly heated brickwork.

A tap-hole for the copper at the lowest point of the hearth and an overflow for the slag at its upper edge complete the essen-

tial features of the fire-brick crucible. Usually, no trapping device is necessary, as will be seen when the management of this type of furnace comes up for consideration. The bottom-plate is hinged, like that of a foundry cupola, the filling-up of the crucible with accretions being one of the most serious drawbacks to the employment of this type of furnace.

2. *The hearth, with blast-trapping device.* — The general style of construction of the hearth depends upon whether the ore-mixture — and its products — is of such a nature that it will tend to keep just about hot enough to maintain the hearth clean and the settler free and liquid, and yet not eat through its enclosing barriers; or whether it will make the hearth too cold, solid slag and matte growing slowly until the interior is blocked nearly to the tuyeres, while the external forehearth — receiving only a sluggish stream of indifferently warm products — freezes even more rapidly still, and soon becomes a monolith rather than a cavity.

Assuming these three typical conditions in sequence, we may first consider the simple case where the hearth and settler are maintained open and hot, and yet where there is no overheating of the bottom-plate or piercing of the walls of the settler or hearth.

The condition of the hearth and settler depends upon the equilibrium maintained between its income and its outgo. The hearth is constantly receiving fresh accessions of heat from the molten products which are showering into it from the smelting zone, and which should be far above the temperature required for their complete liquidity; while it is continuously losing heat in the current of matte and slag which is flowing from it into the settler, as well as from the radiation of heat through its bottom and sides.

When these two sides of the account balance each other pretty closely, the slag will flow free and hot, the hearth and settler will be open and liquid, and the bottom and sides of the interior hearth will retain something like their original form, without either accretions or cavities.[1]

[1] It is better not to complicate this argument by dwelling here upon the modifying effect which may be produced by mattes of different tenor in copper. I will merely say that a matte high in copper (perhaps above 45 per cent) tends to chill and fill up the hearth and settler, while one low in copper (and consequently rich in iron), is extremely corrosive and searching, and will destroy brick or iron if at all superheated.

This is the condition desired under most circumstances, and a matte with about 40 per cent copper, with a slag containing about the same proportion of silica, will, under suitable management, ensure peaceful running, liquid hearth and settler, and freedom from burst-outs.

Under these comfortable conditions, radiation will act automatically and, like the thermostat of an incubator, smooth out the irregularities that arise from moderate fluctuations in the composition of the ore-mixture, or in the quality and distribution of the coke.

If the hearth is running too cold, chilled slag will begin to settle on the bottom, radiation will lessen, and the temperature of the interior will rise. If too hot, the bottom and sides will cut away, radiation will increase as the melted pool approaches the cold bottom-plate, and the hearth will soon return to its normal condition.

If the irregularities become too great, the power of the thermostat will be overtaxed, and the furnace-man must come to its aid, or expect serious trouble — either an outburst of matte, or a chilled settler and hearth.

The means by which this aid may be rendered most efficiently will be considered when the practical management of the furnace is taken up.

It thus becomes plain why the sides of the water-jacket are continued down far below the tuyeres, in some instances, so that they may exert a strong cooling influence upon the hearth; while in others they are stopped as soon as possible, and replaced by brickwork. Also, why the outside settler, which steals heat without giving any in return, must sometimes be omitted and its functions relegated to an interior brick-lined crucible which shall conserve all the heat of the furnace as thoroughly as is possible.

An important adjunct to the hearth, and one which has now reached a highly satisfactory form, is the device for trapping the blast. This attachment is so indispensable to present practice, and is such an advance on our older management, that a few words of explanation are justifiable.

When we first began to appreciate the great advantages derived from allowing our slag and matte to flow together out of the furnace as fast as they became liquid, and of accomplishing

their separation in an outside receptacle of a cheap, accessible, and portable nature, we also began to experience constant difficulty in keeping the blast from blowing out through the same hole whence the molten products came.

This caused a train of evils which are familiar to all furnacemen. Aside from the serious loss of blast, a current of hot smoke and gases issued from the breast-opening with great force, sometimes hurling liquid slag and fragments of glowing coke to a considerable distance, filling the building with dust and unwholesome fumes, and interfering constantly with the desired steady flow of the stream of slag and matte.

The ordinary methods of lessening this evil were two in number, and were both based upon the principle of "trapping" the blast by causing the liquid contents of the interior to rise above, and submerge, the breast-opening, and thus shut off the escape of any gases while the liquids had free egress.

The first method was to close the breast-hole with lumps of clay, and permit the hearth to fill with slag nearly to the tuyeres; then to pierce the clay barrier with a small opening, and allow it to flow off freely. As soon as the level of this accumulated slag had sunk so far as to destroy the trapping effect, the blowing-through of the blast would begin and the dam would be reconstructed. Skilled furnace-men tried at times to regulate the volume of the outflow so as to correspond exactly with the interior accession of melted material; but clay, or even fire-brick, is of too destructible and corrodible a nature to permit of any such delicate regulation. The practical outcome of this plan was that the slag-stream flowed periodically, and that there were forced intervals of cessation. This was highly detrimental to the settler, which chilled during the times of rest and was overloaded during the periods of activity.

The second method of preventing blowing-through was to construct a considerable mound of brick and clay at the breast and, without attempting systematically to raise the level of all the liquid in the interior of the furnace, to foster a sort of local trap which should permit a steady outflow of liquid, and yet prevent the escape of gases. With skilled management, undivided attention, and much hot work this plan achieved at times a fair measure of success, but was one of the most constantly annoying features of the furnace-man's duties.

Recognizing that the principle of trapping the blast by keeping the breast-hole submerged beneath a sufficient layer of liquid products was the sound and simple way of effecting the desired result, it became evident that the key to the problem was the construction of a cheap and permanent dam outside of the breast-hole.

Herreshoff solved it most ingeniously by constructing a water-cooled forehearth which should be an integral part of the furnace-hearth, and which should have a slag-overflow at a certain height that formed the dam in question.[1] This device is admirable, but sacrificed, to some extent, the independence of the forehearth.

With large furnaces and, especially, with products inclined to chill, the smelter demanded a forehearth that should be entirely independent of the furnace, which should discharge its contents into the settler as a faucet discharges water into a bucket. This relegated the dam to the furnace-breast proper, and stimulated invention until a considerable number of excellent trap-spouts have been introduced, and this portion of the furnace equipment is now in a highly satisfactory condition.

At the present time, trap-spouts are standard articles with all the great manufacturers of smelting furnaces and, as such, demand less detailed study in this work than would be the case were the metallurgist compelled to design them for himself. They all depend upon the principle elaborated in the preceding remarks, the variations proceeding from the details of construction.

As an illustration of this important adjunct to every copper blast-furnace, I insert a cut of the Gross trap-spout, which seems to wear well under severe conditions.[2]

As may be seen, it is bolted to the furnace against the face of the little slag-hole jacket which fills the gap in the breast of the great furnace-jacket. It is formed of ribbed, cast-iron plates, bolted together, and lined on the inside with magnesite brick. A lining of this nature is simpler and less inclined to chill than a water-jacketed spout. The wear, in all these devices, comes on the tip where the stream of slag and matte curves over to drop into the forehearth. This tip forms the apex of the dam

[1] See illustration on page 122.

[2] While I am informed that this is a patented article, I must refuse to deprive my readers of useful information for fear of advertising some one's goods.

and has a vertical adjustment of six inches, in order to meet varied demands as to the depth of the trap. It is made of a special copper, is water-cooled, and can be replaced easily.

Hixon, who has given much attention to this matter, makes cheap and satisfactory trap-spouts by casting converter copper around iron pipes, through which his cooling water circulates. These he finds satisfactory in all respects, and gives full description, directions, and illustrations in his book.[1]

Further practical details regarding trap-spouts will be given in connection with the description of typical plants.

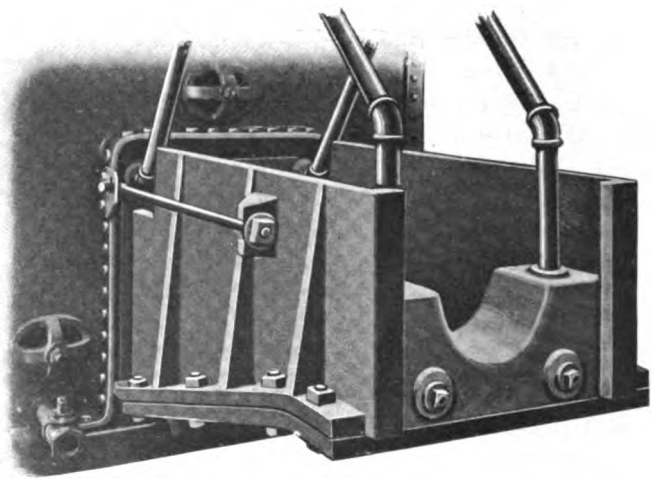


FIG. 33. — Trap-spout

3. *The Forehearth or Settler.* — The transference of certain definite, and almost constant, difficulties from the inaccessible interior of the blast-furnace to an accessible exterior forehearth is one of the greatest advances in the metallurgy of copper.[2]

The external forehearth may be regarded simply as a con-

[1] "Notes on Lead and Copper Smelting, and Copper Converting," by H. W. Hixon, 1908, Hill Publishing Co., New York.

[2] The advantages of the exterior forehearth, or settler, are so universally recognized and adopted that it would not be justifiable to occupy space in arguing upon the subject. In my old "Modern Copper Smelting" (all editions later than 1894) I devote 22 pages to pointing out their value and urging their general adoption. The necessity for such arguments has long passed, but those interested in the subject may find it there fully discussed.

venient temporary receptacle for the stream of slag and matte which flows continuously from the trap-spout of the blast-furnace.

Its duty originally was simple and circumscribed, but, of late years, has become more extended, as it has been found convenient to employ the settler as a sort of connecting link between the blast-furnace and the converter system.

Primarily, its duty is to offer a quiet, sufficiently-heated chamber in which the matte may separate thoroughly from the slag, and this demand implies also, of course, proper means for the removal of these two products after the separation has been effected.

Secondarily, we have come to employ the settler as a reservoir in which matte may be stored up during the intervals in which the converters are occupied, and may supply the molten matte in large quantity when the converter system requires it. This second, and subsidiary, duty has led to an enlargement of the forehearth far beyond the size required for separating purposes, settlers being now in common use of 15 ft. (outside) diameter, and with a capacity of 40 tons (36.3 m.t.) of matte, and more.

The duties which are demanded of the forehearth indicate clearly the principles which should underlie its construction.

Reduced to their simplest form, we see that there is required a convenient receptacle standing in a position to receive the stream of slag and matte which falls from the trap-spout of the furnace. In order to conserve heat and prevent chilling, its walls must consist of material which shall not conduct heat too rapidly, while, on the other hand, they must be capable of resisting the action of the liquid contents in cases where the furnace is running unusually hot and fast, or where the matte or slag are of a corrosive nature.

As these conditions are most satisfactorily and economically fulfilled by the ordinary refractory materials (fire-brick, clay, etc.), it becomes necessary to hold them together with iron bands or plates; and as the products arising from the melting of the ore are flowing constantly into the settler, provision must be made for their continuous, or periodic, removal.

With a clear understanding of these demands, any person who is accustomed to the handling of melted ores could construct a settler which would effect the desired result after some

fashion; but the experience of a generation of metallurgists has detected and corrected the weak points, and a brief description of the present status of the forehearth will obviate all necessity for experimentation.

Considering, in the first place, what would naturally be the most vulnerable portions of the settler, we turn our attention at once to those points which are required to withstand the dissolving and wearing effects of a flowing stream of matte or slag. Under ordinary conditions, a slag-spout can be made of nothing cheaper and better than cast-iron. This material withstands satisfactorily the chemical and mechanical effect of the slag, and is strong enough to resist the customary violent punching of the attendant. Certain exceptional slags (especially when highly ferruginous, and containing considerable quantities of sulphides in solution) cut a deep groove quickly into the bottom of the spout. This may generally be remedied by lining the spout carefully with clay, and permitting it to dry for a short time. If this proves ineffectual, a small water-jacketed copper tip — or some similar contrivance — will obviate the difficulty.

The tap-hole for the matte is situated near the bottom of the forehearth. In large settlers, duplicate tap-holes are provided, and, instead of simply cutting a hole in the encircling iron shell which shall correspond to a similar aperture in the brick lining and filling this in with brick and clay, a more elaborate system is pursued. The objection to the method just described is that a clay-and-brick tap-hole is a shiftless and unsatisfactory device — for reasons known to all furnace-men — and is unsafe as a gateway for a large volume of matte.

Under ordinary conditions, a heavy cast-iron tap-block is satisfactory, as the issuing stream of matte does not flow for a period of time long enough to destroy it. The tap-hole thus becomes a circumscribed aperture at a fixed level, and is easily opened and closed.

Where the matte is of so destructive a nature that even this device is inadequate, a water-cooled casting of copper may be substituted; or — as used by Hixon — a block of converter copper cast around iron pipes through which water is circulated.

The smaller forehearths are often made portable and mounted upon wheels, for convenience of changing. While occasionally water-jacketed — see p. 122 — they usually need all of their

available heat to prevent their contents from chilling, and may consist of cast-iron plates bolted firmly together and lined with refractory material. Where the slag is silicious and the matte scanty and non-corrosive, the forehearth is more likely to chill than to cut. In such cases it is idle to waste choice refractory material in the settler. Anything to protect the iron envelope from direct contact of slag and matte will answer. Broken fire-brick, red brick, or even a plaster of clay and horse manure will suffice.

The accompanying illustration, Fig. 34, shows a convenient forehearth of the portable type, and requires no particular descrip-

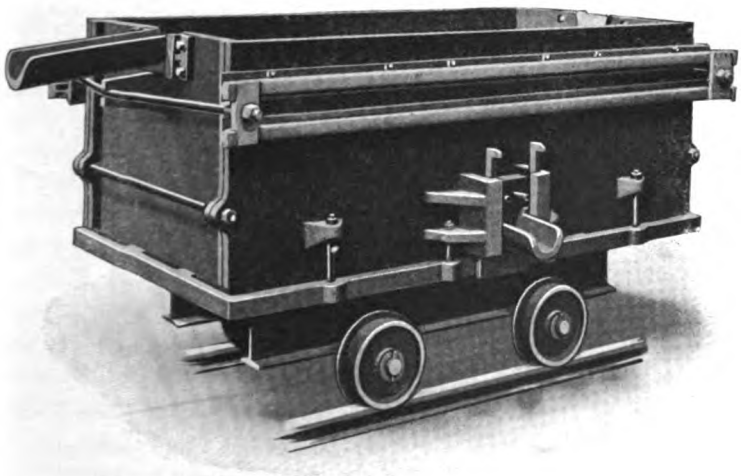


FIG. 34. — Portable forehearth

tion. An extra forehearth is usually provided so that the change can be made without delaying the furnace. The old forehearth, tapped dry, is run to one side and cooled sufficiently to permit the removal of the side-and-end-plates. The chilled block of accretions is tumbled to one side and allowed to cool, prior to breaking-up and re-smelting, while the iron plates are again set up and relined for service.

The matte tap fitted to this settler is a heavy block of cast-iron, with a cooling-pipe cast inside around the tap-hole. Both this block and its spout are adjustable vertically, and are held in place by two gib-head steel wedges.

The Orford siphon-tap [1] forehearth is an outside settling device so arranged that the matte and slag are discharged from it in separate and continuous streams. See Figs. 35, 36.

It consists of a rectangular box, some 5 feet by 5 feet 6 inches, formed of cast-iron plates strongly bolted together at the corners, and lined with a brick wall 4½ inches or 9 inches thick, according to the quality of the product. It is fastened firmly to the front of the furnace, and is divided longitudinally by a 9-inch wall of

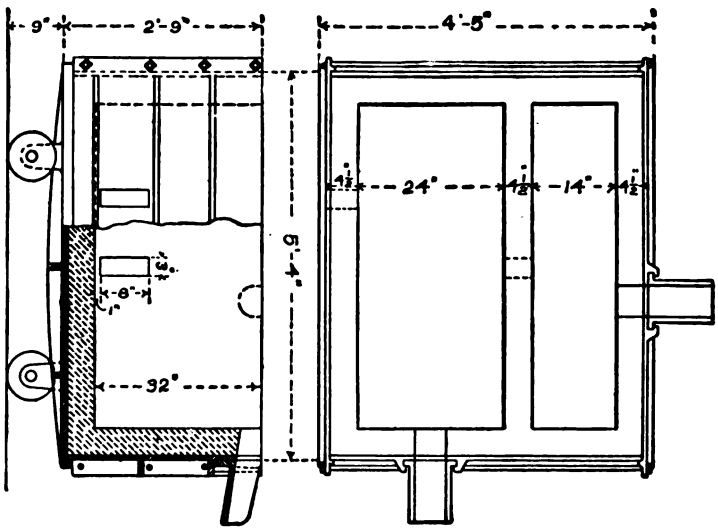


FIG. 35. — Siphon-tap forehearth — plan

FIG. 36. — Siphon-tap forehearth — longitudinal section

fire-brick into a greater and lesser portion, the area of the two compartments being about as five to two.

The entire molten contents of the furnace discharge into the larger of these two compartments, which is provided with a slag-spout bolted to the upper edge of the front plate, while it communicates with the smaller compartment by means of a 3-inch by 8-inch vertical slot through the 9-inch division wall, about midway of its length and on a level with the floor of the forehearth. This smaller compartment also has a spout about

[1] This is a misnomer, as the apparatus here referred to, as used for the continuous discharge of the metallic product, has nothing about it pertaining to the principles of the siphon.

2 inches below the level of the spout belonging to the larger division, and on the outer side, instead of the end wall, for the sake of convenience.

A thorough understanding of this simple and inexpensive contrivance will render it easy to appreciate its management.

When the breast-hole is opened, and slag and metal first begin to flow, the larger compartment is soon filled, as the only means of communication between the two divisions of the forehearth is the closed slot in the lower part of the 9-inch division wall.

The molten products separate according to the law of gravity, and slag is allowed to flow through the spout of the large compartment until the drops of metal appearing show that it is filled with the more valuable product. The channel of communication is now opened by means of a crooked tapping-bar, and the metal flows rapidly through the same into the smaller compartment until an equilibrium is established and both divisions of the forehearth are partially filled with the matte, the communicating channel being far below the surface of the same, and consequently so situated that slag can never reach it unless it should sink below the metal, which is obviously impossible.

As the furnace constantly discharges its stream into the larger compartment, the forehearth is soon filled again, the metal sinking to the bottom and standing at the same level in both divisions, while the slag simply flows over the surface of the matte in the larger compartment.

As soon as the matte reaches the level of the spout attached to the small compartment, it begins to flow into a pot placed to receive it, and by judicious manipulation, and if a sufficient proportion of matte is produced from the charge, a constant stream of each product may be kept running without difficulty.

The management of this siphon-tap requires considerable experience, as the matte stops occasionally without apparent cause, and requires a certain amount of manipulation and coaxing to keep running freely. This is accomplished by slightly damming up the slag-spout, which soon forces an excess of matte into the smaller compartment, or by clearing out the communicating orifice by means of a heated bar bent to the required curve.

With matte of 50 per cent or over, the principal difficulty is found in the gradual filling up of the forehearth by chilling, while the matte containing 20 per cent or less of copper, and produced

in large quantities, has directly the opposite effect, thinning the brick lining until the plates are endangered, and cutting away the division wall until the two compartments are virtually thrown into one.

But even under these circumstances, and so long as a vestige of the center wall remains, the separation of the matte and slag continues to be perfect, and by judicious repairing and nursing a forehearth apparently in the last stage of ruin may yet do good service for many days.

An opening through the division wall 18 inches high by 24 inches wide, and actually involving two-thirds of the separating brickwork, is not incompatible with a perfect separation.

The larger compartment is provided with a tap-hole at its lowest boundary, and on the side opposite the matte division, and a quantity of sand should always be at hand ready to make up into rough molds in case of any sudden necessity for tapping.

This same idea of a matte-separating forehearth, which originated at the Orford works in the shape of the small apparatus just described, has been amplified, and adapted to present conditions, by W. J. Watson, manager of the Tyee smelter.[1]

This is an adaptation and improvement of the old Orford separator, in which the receiving compartment is very large and is provided with a water-jacket, while the small matte compartment is placed outside of the main settler, so that in case of accident it is practicable to remove the matte-annex without interfering with the working of the furnace. In such emergencies the matte is tapped in the customary fashion, the main settler being provided with an ordinary tap-jacket on either side.

The matte compartment, as shown in the accompanying illustration, is 24 in. (0.61 m.) square and is lined with eight inches (0.203 m.) of brick. The opening between the main settler and the annex is very large — 22 × 24 in. (0.056 × 0.61) — but with the exception of a connecting channel six inches (0.152 m.) square, at the bottom, is filled in with chrome brick.

The level of the matte is controlled by regulating the height of the slag outlet, an average depth of 10 in. (0.254 m.) of matte in the settler being desired.

The size of settler ordinarily used at these works has been

[1] I make free use of an illustrated description of this valuable device, published by E. Jacobs in the *Eng. and Mining Journal* of June 19, 1909.

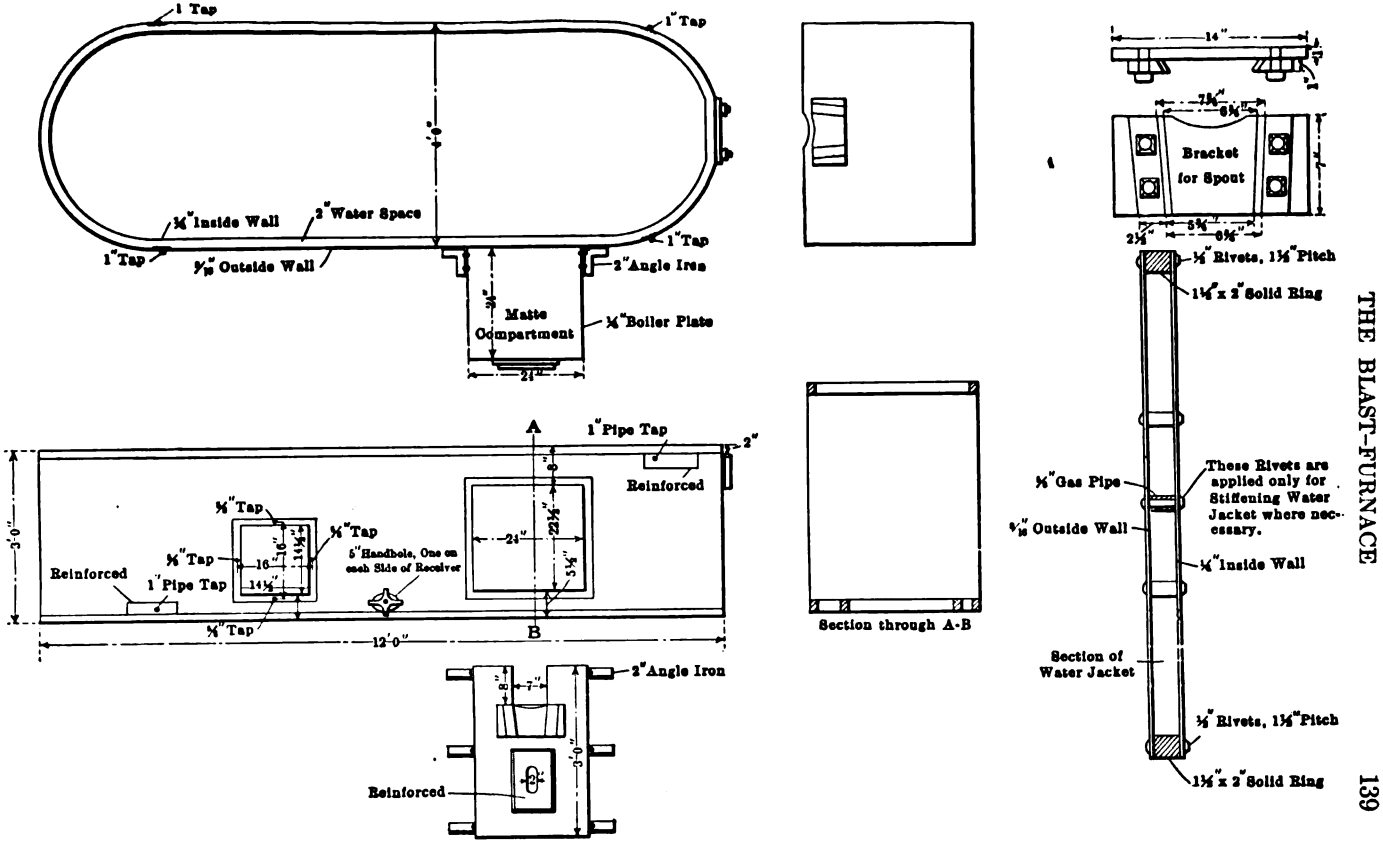


FIG. 37. — Tyee forehearth

4 × 12 ft. (1.22 × 3.66 m.), but the construction of a much larger one is contemplated. During the three years which this settler has been in use, it has been entirely satisfactory, the matte compartment having chilled only two or three times, and then from an excessive amount of ZnS in the matte — reaching at times 23 per cent.

Among the advantages experienced are the elimination of danger to the men while tapping, the much cleaner slag, the saving of a tapper on each shift, the reduction in wear and tear of matte pots (the stream of matte not striking the side of the pot as in the ordinary tapping operation), and the saving of tapping clay. I have no doubt that many blast-furnaces would benefit by the introduction of a device of this nature.

The large, circular forehearths may be seen in succeeding cuts, and will be described in connection with the plants to which they belong.

In the *Engineering and Mining Journal* of Oct. 14, 1905, there is a most instructive paper by Hiram W. Hixon, entitled "The Evolution of Tap-Jackets and Spouts." I refer to it in connection with devices which may be useful to metallurgists who are troubled with exceptionally corrosive mattes which destroy any of the ordinary tap-spouts or jackets.

In about 1884 I erected two 100-ton Herreshoff water-jacket furnaces for the Canadian Copper Co. at Sudbury, Ontario. The nickel-copper matte, which we were producing from tolerably rich ores, was so destructive of our tapping-blocks as to cause much delay and inconvenience. The jacketed connection between furnace and forehearth lasted pretty well, with the comparatively slow running of those days, but the matte-tap from the forehearth was destroyed almost as fast as we could replace it. Iron, copper, bronze, and various other alloys were tried for the jacketed casting which formed the tap-block, but were all unsatisfactory until one of the machinery firms supplied us with a cast jacket of pure copper to which had been added a small percentage of silicon. This ensured a casting without flaws or weak spots and, although far from permanent, enabled us to get along without prohibitive delays and costs.

With the rapid smelting of later times, these same troubles were increased, and Hixon speaks of the serious difficulty that he experienced in devising a suitable trap-spout for the furnace,

and a reasonably permanent tap-block for the settler. He says: "We now use a water-jacketed box, lined with chrome brick, provided with a bronze overflow. This box is built in front of the discharge from the furnace, and the overflow is higher than the inlet from the furnace. The matte tap from the settler has passed through similar development. Copper blocks with and without water circulation, cast-iron blocks with pipe coils for water circulation, and cast-bronze jackets have all been used. Cast-iron blocks without water circulation are in use by the Canadian Copper Company. The hole burns out rather large, and it is necessary to use a conical clay plug (about 6 in. in diameter at the large end) on a stopper handled by two men from behind a shield having a hole through it large enough to pass the plug through. When the ladle is full the two men throw all their weight on the stopper, guiding it into the tap hole, and a third man drives it in with a sledge. If they fail to make a stop a second stopper is provided, and if that fails everybody seeks a safe place until the settler has emptied itself on the floor. I have used copper blocks cast around iron pipe, chrome blocks without water circulation, but have the best results from carbon blocks made for electric smelting electrodes. A 2-in. hole is bored through the block, which is 5 in. thick, and a cast-iron plate with a hole to match is placed on the outside of the carbon block to strengthen it. The matte stream passing through the carbon block does not act on it as rapidly as on cast-iron or chrome, and it has the advantage that it does not cause explosions as a water-cooled jacket when it burns out. Copper-nickel matte is much more severe in its action on spouts and jackets than copper matte, and all available material has been tested in the construction of spouts for its conveyance."

While the principal portions of the water-jacket blast-furnace have been shown to be simple and intelligible units, and while they have become standard articles of production, their correct construction is based upon long experience and a multitude of trials and failures. As is the case with all forms of highly specialized apparatus, each one of their many details has been worked out with the most anxious care, and their makers must have a deep fund of experience upon which to draw before their products can be accepted as safe and satisfactory.

In view of the severe duty which it is called upon to perform, the manufacture of the jacket itself is the most critical and important part of the entire operation; and, while it would be out of place in a book of this scope to occupy much space in describing the technical details of manufacture, it is important that the metallurgist should have sufficient general knowledge of the subject to appreciate the peculiar care and skill that are involved in the construction of a high-class jacket, and to be in a position to judge whether his order for a furnace is being executed in a proper manner.

The material which has been found most suitable for the construction of the water-jacket is the choice grade of boiler-plate known as "flange-steel," and should have a tensile strength of some 60,000 lb. per sq. inch.[1]

The method of manipulating this flange-steel varies somewhat with the different makers, and it is manifestly impossible to describe all of the devices practised by the principal manufacturers. I shall, therefore, cite a single illustrative method, and describe briefly the manner in which water-jackets are constructed at one of the large furnace-building plants in the United States.

One of the main points of a good jacket is that all weaknesses, of whatsoever description, shall be eliminated from its inner surface, so that where it comes in contact with the actual smelting there shall be a smooth, unweakened sheet of metal, presenting no point of attack to the destructive influences which prevail within the furnace shaft. Consequently, rivets, seams, joints, welds, etc., must be eliminated from the inner surface of the jacket, and transferred to the exterior, where they are not only comparatively free from harmful influences, but can easily be dealt with if they develop leaks or other imperfections. An appreciation of this point will explain certain costly elaborations in their manufacture which would, otherwise, appear superfluous.

After the sheet of boiler-plate has been cut to the required size, its sides and edges are turned up at right angles, or "flanged." This is effected by means of a flanging machine of great power, the steel sheets used being, perhaps, $\frac{3}{8}$ in. thick for the inner, and $\frac{5}{16}$ in. thick for the outer plate (0.0096 to 0.0047 m.). The water-space is four to five inches wide (0.102 to 0.127 m.).

[1] With a view to avoiding the destructive effect of sulphates in certain moist ores, copper is sometimes employed for the inner sheet of the jacket.

With the bending of the sides and ends of a stiff sheet of any material, there arises a complication at the corners, as may be illustrated by folding up the four edges of a sheet of paper for about half an inch and noting that, when these folds are maintained at a right angle to the sheet, the corners will flare out in a curve. This difficulty is met, often, by cutting out the excess of metal and then welding the corners.

The method which I am describing, however, involves a more expensive plan of dealing with the excess metal at the corners. The plate is heated and held upon a cast-iron form having the shape of the inside of the corner of the jacket, and is hammered down gradually with sledges in such a manner that the plate — while being shaped to the form — is not allowed to buckle or fold upon itself. This is accomplished by forcing the metal in upon itself and actually increasing its thickness, as in "upsetting" the end of a steel rod. By this method welds are completely avoided, and all rivets are exterior and accessible.

The tuyere opening through the jacket demands careful attention to prevent leakages, and may be constructed in various ways. An excellent plan is to form the tuyere opening with a forged-steel tapering thimble, about $\frac{3}{8}$ in. (0.0096 m.) thick. This is, of course, set within the water-space, and has a shoulder turned down on its inner, or smaller, end to set against the water-surface of the inner sheet. The projecting end is beaded over the fire-sheet like a boiler-flue, while the outer end of the thimble is riveted to a flanged nozzle in the outer sheet. This construction avoids all interior rivets, and "stays" the jackets at a point where it is most needed, while other stays (all without inside rivets) are used to stiffen the larger jackets.

Fig. 38 shows the fire side, and Fig. 39 the exterior of such a water-jacket, with particular reference to the tuyere-opening as just described. The exterior view shows in addition the eyes by which the blast-pipe is held against the tuyere-opening, as well as the hand-holes and blow-off openings for cleaning out the water-space.

In furnaces of moderate size, the regular discharge of slag and matte takes place at one of the ends of the furnace, and the corresponding end-jacket is provided with an opening for the insertion of the "tap-jacket." This is simply a small, easily removable jacket, designed particularly with a view to with-

standing the wear of the great volume of superheated slag and matte which must pass through its orifice. It is often of cast copper, drilled and tapped for the inlet and overflow pipes of the cooling water. The trap-spout — already described — is bolted up against the furnace directly opposite the orifice of the little tap-jacket, as may be seen in several of the illustrations.

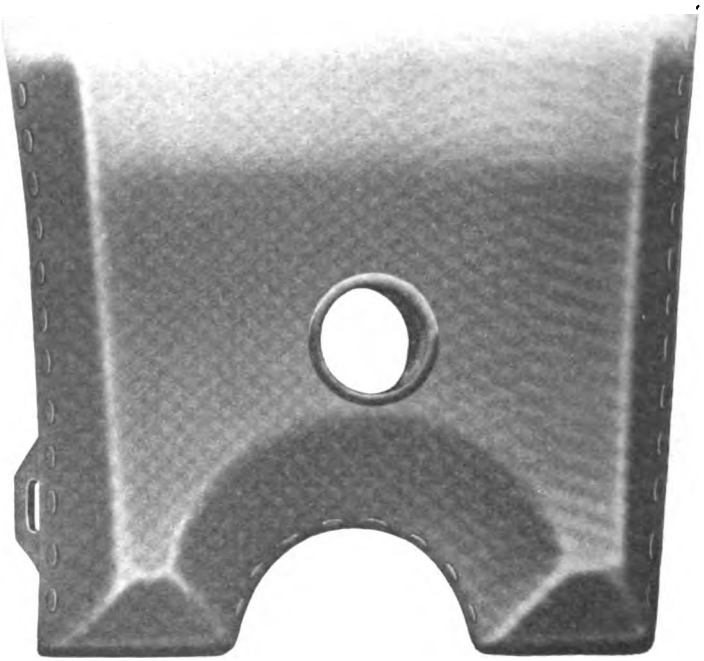


FIG. 38. — Fire side of jacket

The end-jackets are made perpendicular, while the side-jackets are usually boshed — beginning at a point not far above the tuyeres and continuing throughout their entire length, the perpendicularity of the shaft-interior being resumed above by the brickwork, or by a second tier of jackets placed vertically. The amount of bosh is an important factor in smelting, and is determined by the quality of ore and fuel, as well as by the results at which we aim; this point will be considered later.

The actual dimensions of many jackets — as well as those of the furnaces of which they form a part — are given in the fol-

lowing table, which also contains the other important dimensions of many of the principal furnaces of North America.[1]

In cases where metallic copper, abnormally high-grade matte, silicious slags, or slow smelting preclude the employment of a distinct outside settler, the furnace is arranged with an interior crucible in which the separation of metal and slag takes place.

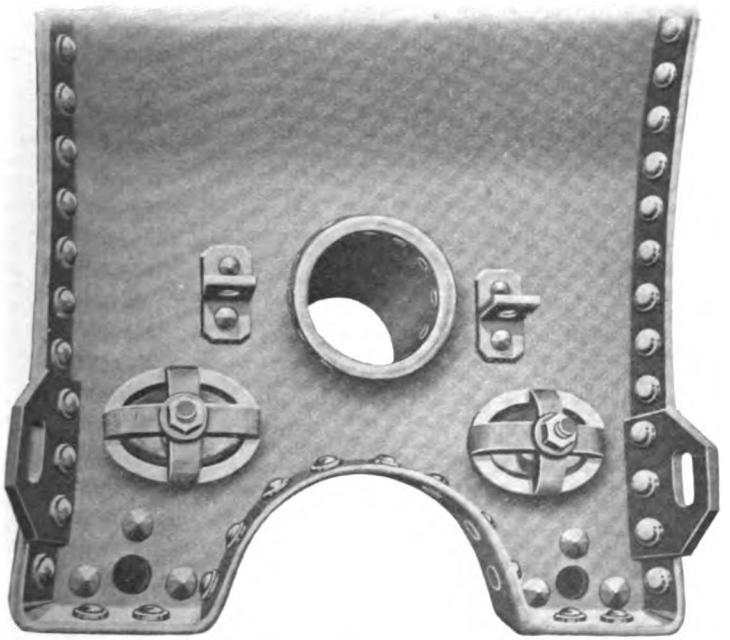


FIG. 39. — Exterior of jacket

The metal is tapped off as it accumulates, and the slag is either tapped periodically and run through a large settling-pot, or — if the furnace has a sufficient capacity — it may be equipped with trap-spout and a regular forehearth. In any case, careful exterior settling of the slag is necessary, even though the inside separation is good. Globules of matte or copper are sure to come over with the slag to some extent, and the high grade of the metal accentuates the loss.

[1] This table was compiled and prepared by Mr. H. B. Lowden, of Denver, to whom I am still further indebted for aid and information.

PRINCIPAL DIMENSIONS OF CER

			Date	Size in Inches	Hearth Area in Square Feet
D1	North American Smelter & Mines Co.	Golden, Colo.	1901	36×192	48.00
D1	" " " " " "	" "	1902	42× 80	23.33
	Mazapil Copper Co.	Concepcion del Oro Zac	1903	38×120	31.67
	Lost Packer Mining Co.	Ivers, Idaho	1905	40× 96	26.66
2	Teziutlan Copper Co.	near Teziutlan, Puebla	1903	45½×120	38.13
	" " "	" " "	1909	54×240	90.00
D	Soc. Beneficiadora de Tocopilla	Tocopilla, Chile	1907	38× 84	22.17
D	Bully Hill Copper Mg. & Smelting Co.	Winthrop, Calif.	1903	42×120	35.00
G	Silver Fissure Mining Co.	Polaris, Mont.	1905	36×108	27.00
	Cananea Cons. Copper Co.	La Cananea, Son.	1908	48×210	70.00
	United Verde Copper Co.	Jerome, Ariz.	1907	48×240	80.00
	Rocky Mountain Smelting Co.	Florence, Colo.	1900	36×180	45.00
D	Robinson Gold Mining Co.	New Zealand	1907	33×144	33.00
	New World Smelting Co.	Gardiner, Mont.	1906	42×120	35.00
	Furukawa Mining Co.	Ashio Smelter, Japan	1907	42×160	46.66
	Balaklala Copper Co.	Coram, Calif.	1907	56×240	93.66
	El Cobre Mines	Santiago de Cuba	1904	40×160	44.39
	Saddle Mountain Mining Co.	Christmas, Ariz.	1907	44×200	61.10
	Engels Copper Mining Co.	Taylorsville, Calif.	1909	40× 96	26.66
D	Modern Smelting & Refining Co.	Modern, Colo.	1909	38×180	47.50
	Anaconda 87 ft. Washoe	Anaconda, Mont.	1906	56×1044	406.00
C	New York & Nevada (Nev. Cons.)	Ely, Nevada	1902	42×120	35.00
	Granby Cons. M. S. & P. Co.	Grand Forks, B.C.	1909	48×260	86.66
	" " M. S. & P. Co.	" " "	1909	44×266½	81.41
	Copperfield Mines	Copperfield, Vt.	1901	38×144	38.00
	Giroux Cons. Mines Co.	Ely, Nevada	1905	42×120	35.00
C	United Globe Mines	Globe, Ariz.	1900	42×144	42.00
	Copper Crown Mining Co.	Pictou, Nova Scotia	1901	36× 60	15.00
D3	Mountain Copper Co.	Keswick, Calif.	1896	42×150	43.75
	Cia. Metalurgica Nacional	Matehuala, S. L. P.	1905	42×120	35.00
C4	Montana Ore Purchasing Co.	Butte, Mont.	1899	40×120	33.33
	Cataract Copper Mining Co.	Basin, Mont.	1903	40×144	40.00
	Amer. Smelting & Refining Co.	Aguascalientes	1901	44×150	45.83
	Grand Prize Copper Co.	Jerome, Ariz.	1901	36× 90	22.50

A — Two dimensions indicate two tiers of jackets. Height of lower tier given first.
B — Jackets straight (without knee bosh).
C — For smelting oxide ores to black copper or for inside separation.
D — Hot blast.

TAIN COPPER BLAST FURNACES

Centers of Tuyeres to Feed Floor	Centers of Tuyeres to Tapping Floor	Height of Jackets in Inches	Centers of Tuyeres to Lower Line of Bosh. Inches	Centers of Tuyeres to Upper Line of Bosh. Inches	Centers of Tuyeres to Lower Edge of Jackets	Bosh in Inches	Width of Lower Side Jackets. Inches	Number and Diameter of Tuyeres	Center to Center of Tuyeres	Total Area of Tuyeres in Square Inches	Sq. In. Tuyere Opening per Sq. Ft. Hearth Area
		A									
16′ 5″	6′ 3″	102+36	6″	75	27	13	48	32, 5″	12″	628.2	13.80
17′ 0″	5′ 8″	72+46	6	58	14	12	40	12, 5″	12½	235.6	10.10
12′ 0″	6′ 3½″	114+36	B	120½	30	6½	30	16, 4½″	15	254.4	8.03
10′ 11″	6′ 9″	96+48	6	66	30	12	48	16, 4½″	12	254.4	9.54
7′ 9″	4′ 5″	120	B	93	28½	7½	40	18, 3″	13	127.3	3.34
14′ 6″	8′ 6″	120+66	B	140	46	12	48	40, 5″	12	785.3	8.73
10′ 6″	6′ 6″	144	6	66	30	12	42	12, 4½″	14	190.8	8.56
9′ 1″	5′ 10″	96+27	6	66	30	9	40	18, 4½″	13½	286.2	8.18
20′ 0″	7′ 0″	138	6	75	28	12	48 & 60	18, 4½″	12	286.2	10.60
10′ 4½″	—	150¾	B	115	36	9	—	36, 4½″	11⅜	637.9	9.11
10′ 1″	8′ 11″	72+48	B	61	11	10	120	40, 5½″	12	1038.8	12.99
13′ 9″	6′ 3″	104+36	6	77	27	13	60	30, 5″	12	588.9	13.09
11′ 4″	6′ 5″	96+48	6	66	30	12	48	24, 4½″	12	381.6	11.56
11′ 0″	6′ 9″	96+48	6	66	30	12	40	18, 4½″	13½	286.2	8.18
11′ 4″	6′ 8″	96+48	6	66	30	10	53 5/16	24, 4½″	13⅜	381.6	8.18
18′ 10″	8′ 2″	122+96	—	—	42	—	—	42, 4″	11½	507.9	5.42
11′ 3″	6′ 9″	96+48	6	66	30	12	40	24, 4½″	13½	381.6	8.82
9′ 1″	6′ 5″	96+48	6	66	30	12	50	32, 4½″	12½	508.8	8.33
10′ 6″	6′ 0″	96+42	6	66	30	10	48	16, 4½″	12	254.4	9.54
12′ 10″	7′ 9″	96+36	6	66	30	12	60	30, 4½″	12	477.0	10.04
19′,0″	7′ 9″	90+90	B	78	11 & 17	8	84 & 90	150, 4″	12 to 14	1885.0	12.57
14′ 4″	5′ 8″	102	5	58	14	10	40	18, 4½″	13¼	286.2	8.18
12′ 10″	5′ 2″	97½+88	—	—	34	—	52	60, 3½″	8⅛	577.2	6.66
12′ 10″	5′ 2″	90+53	22	56	34	12	53	30, 5″	17¾	589.0	7.23
10′ 8″	6′ 3″	102+30	6	75	27	11	48	24, 4″	12	301.68	7.94
11′ 0″	5′ 9″	96+48	6	66	30	12	40	18, 4½″	13½	286.2	8.18
—	5′ 9″	68+24	14	60	9	12	48	18, 4½″	16	286.2	6.81
8′ 9″	5′ 2″	108+27	6	70	30½	6	60	10, 4″	12	125.7	8.38
9′ 3″	4′ 8″	53+70	15	33	20	6	30	24, 3½″	15	230.88	5.28
11′ 0″	5′ 9″	96+48	6	66	30	12	40	18, 4½″	13½	286.2	8.18
11′ 2″	4′ 10″	54+36+36	6	45	9	12	15	16, 4″	15	201.12	6.03
9′ 6″	6′ 9″	122	6	59	27	10	48	24, 4½″	12	381.6	9.54
11′ 1″	5′ 3″	114	B	82	32	4½	30	20, 4½″	15	318.0	6.94
8′ 1″	6′ 10″	84+40	6	57	27	12	90	14, 4″	12½	175.98	7.82

All jacket-connections are held together with heavy wrought-iron lugs and rods at each joint, and the lower ends of the bottom tier of jackets are confined firmly by the bottom-plate.

Fig. 41 is from a photograph of the lower corner of a furnace, showing details of bottom-plate, and of lower portion of jackets.

R is an end-jacket; S, one of the side-jackets; Q, the bottom-plate; K, method of fastening end-jacket to end side-jacket; N, hand-holes (two at bottom of each jacket); M, blow-off (two for each jacket); O and P, devices for maintaining the jackets accurately in position.

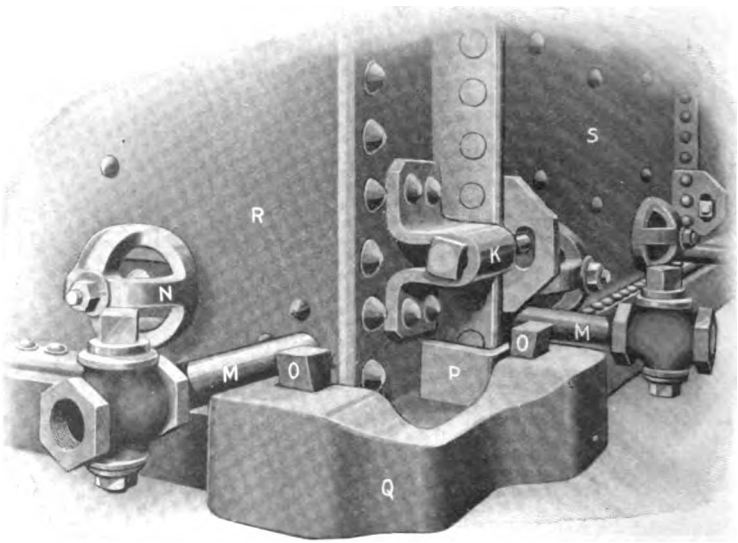


FIG. 41. — Lower corner of furnace

Fig. 42 is from a photograph taken to show further details of the jackets, and especially of the tuyeres, blast-pipes, and connections.

A is a tuyere, held against face of jacket by hook-bolts. In the face of the tuyere is a groove, filled with asbestos backing, which makes a close connection with the projecting flange of the tuyere-thimble. B, stuffing-box forming a flexible connection between tuyere and blow-pipe. This connection may be placed above, where the blow-pipe takes off from the bustle-pipe. All sharp bends between bustle-pipe and furnace should be avoided.

C, quick-opening gate-valve built into the tuyere. D, nipple fitted into a taper hole. Screw-threads are eliminated to avoid delay when barring tuyeres. E, tuyere-cap held to tuyere by

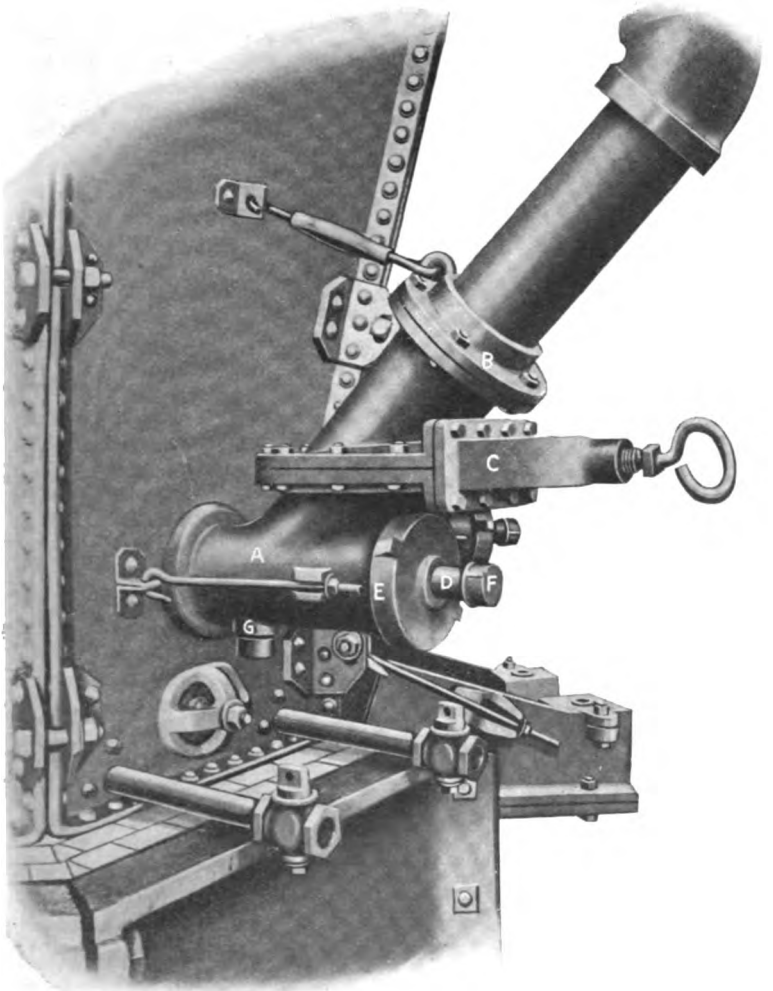


FIG. 42. — Water-jacket furnace tuyere-connections

an interrupted screw for quick removal. F, cap on end of nipple, with very small peep-hole tapered out large inside. G, slag-escape; this is intended merely as an indicator; the cap covering

the nipple is pierced with a large hole, and the escape of wind is prevented by dropping in a pasteboard disk. The tuyere can be almost instantly removed.

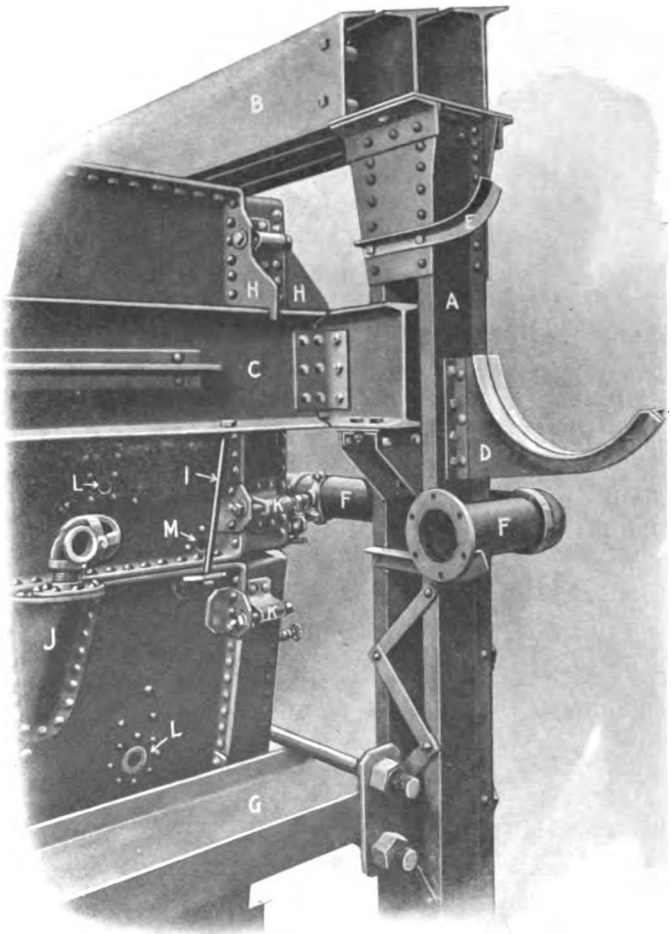


FIG. 43. — Water-jacket furnace. Method of supporting jackets

Further details of water-jackets and their connections are illustrated in Fig. 43, the photograph for which was taken especially to show the manner of supporting a double tier of jackets. In order not to obstruct the view of these details, it was necessary to remove the bustle-pipe, waste-water gutter, jacket-water supply

pipe, and a portion of the mantel frame. The general arrangement of the parts which are missing in this cut may be seen in Fig. 44, page 152.

A is a main corner column; B, the mantel frame of I-beams and channels. The space between inner edge of channels and inside of top of upper jackets is covered by cast-iron bevel feed plates, which are not shown. C, I-beam frame resting on brackets on corner columns, to carry jackets; D, bracket to carry bustle-pipe; E, bracket to carry waste-water gutter; F, jacket-water supply pipe; G, I-beam binder-frame, to take outward thrust of lower jackets; H, means for carrying upper jackets on frame C; I, method of hanging lower jackets from frame C; J, pitcher-spout discharge neck to prevent trapping steam; E, lugs for binding jackets together; L, feed-opening for jacket-water; M, opening tapped for blow-off.

The succeeding illustration (Fig. 44) shows a fully equipped furnace of 44 × 200 in. (1.12 × 5.08 m.) hearth area. It has 32 tuyeres and two tiers of steel-plate water-jackets. The trap-spout is a prominent object in the illustration, and the tap-hole for emptying the hearth for repairs may be seen at the middle of the long side. The mantel and flue-connections are not shown, nor is the forehearth indicated.

The need of a sectionalized furnace for mountain transportation is often experienced in remote districts, and manufacturers have expended much ingenuity in devising suitable apparatus for this purpose. Even a small, circular, self-contained jacket is too heavy and unmanageable for the only form of transportation available in certain instances and, moreover, is difficult to clean out and costly to repair, when experiencing the mishaps inseparable from producing metallic copper or rich matte on so small a scale and with inefficient labor.

Fig. 45 illustrates a small, approximately circular, furnace which contains no piece weighing more than 325 lb. (147.4 kg.). It is composed of 10 narrow water-jackets, and is equipped as completely as the larger furnaces. It may, of course, be run either with internal crucible or with an outside separator, although its small stream of melted products demands a modified treatment of the latter.

Personally, I should prefer the furnace illustrated in Fig. 46, which also is sectionalized, having no piece exceeding 325 lb. (147.4 kg.) in weight, but is rectangular in form, with a section at the tuyeres of 33 × 36 in. (0.838 × 0.914 m.).

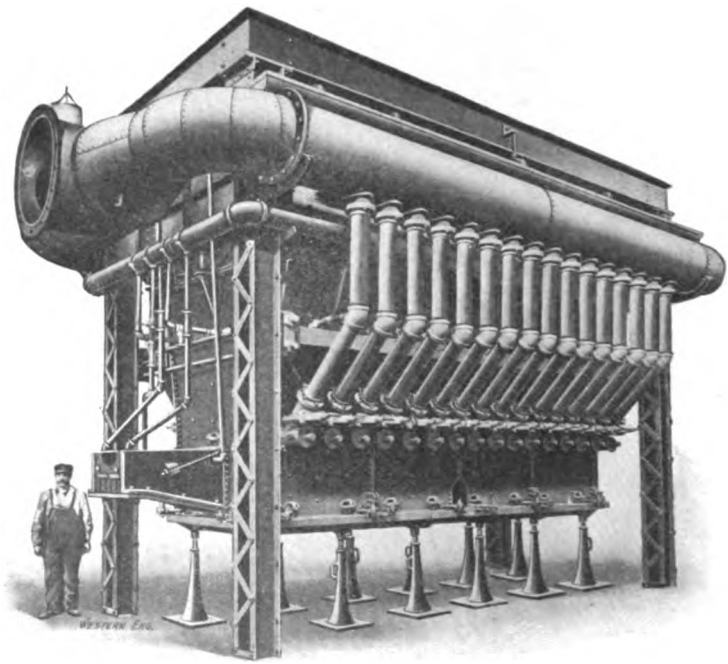


FIG. 44. — Copper-matting furnace

Its details are worked out with much care so as to permit of stopping, clearing out, and starting again with expedition and economy. This is effected by suspending the jackets by chains from the I-beams which rest upon the corner columns, and by having a portable crucible supported by jack-screws, and arranged so that it may be lowered upon a truck, and replaced by a duplicate.

In case of a freeze-up, the crucible is lowered upon its truck, run to one side, and the furnace opened in the following manner: The T-iron binders are taken down, the bolts connecting the jackets on one side are removed, and the various connections severed. The jackets, being flexibly supported by chains, may

now be swung to one side without taking them down, and the interior of the shaft is completely accessible.

The cooling water is usually delivered to the jacket at, or near, its upper end and, being heavier, sinks and displaces upward

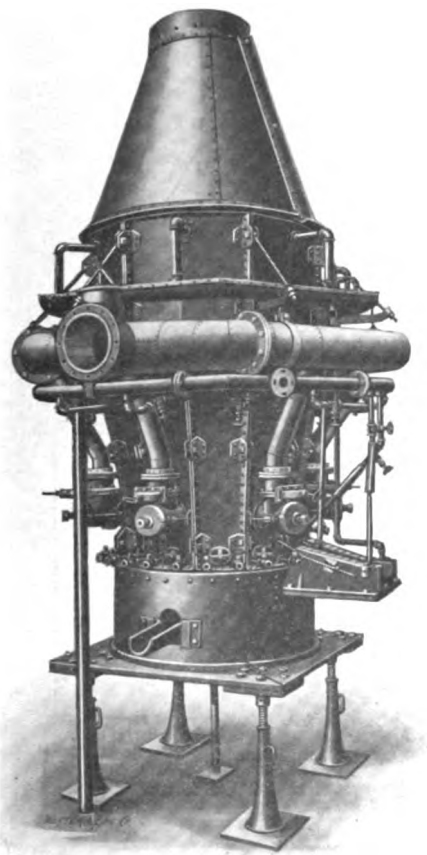


FIG. 15. — Sectionalized 10-sided furnace

the warmer water, which is drawn out at the highest point of the water-space, especial care being taken to permit no space for the collection of steam.

The amount of cooling water required for a jacketed furnace of a given size varies so excessively that any attempt to give average figures would be misleading. It varies with the height

of the jackets, the quality and quantity of the fuel, the fineness and height of the ore-column, the pressure of the blast, the character of the slag, the tendency of the ore to form wall-accretions, and many other factors.

Of course, during the blowing-in operation, where there is an abnormal proportion of coke, and a naked metallic surface not

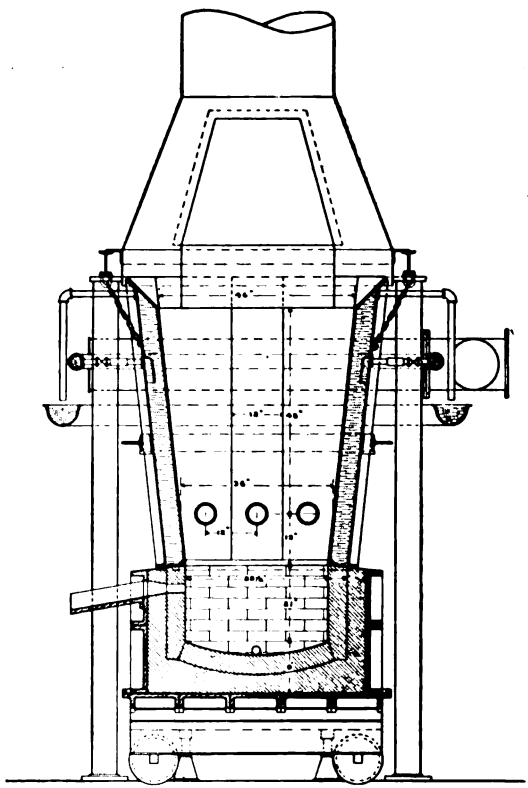


FIG. 46. — Sectionalized water-jacket furnace — rectangular

yet protected with a comparatively non-conducting layer of ZnS, or other volatile constituents of the charge, the consumption of cooling water is excessive; so, also, when the furnace is running badly and the fire is diffused throughout the entire ore-column. On the other hand, when the smelting zone is concentrated where it belongs, the top cool, and the jackets coated, the consumption of water may sink to one-fourth of the former amount.

Where water is scarce, it may be cooled and re-used. Dripping through high towers of brush, or some device of this kind where great surface is exposed to the air, is much more effectual in cooling the water than the same value expended in sluices or zigzag boxes. Speaking solely from personal experience, I believe that in a very dry climate (Arizona), and with reasonable appliances, there will be a loss of about one-fourth of the water each time that it passes through the jackets and undergoes the cooling necessary for its use for the second time.

Furnace-men are in the habit of using too much jacket-water, finding it less trouble to turn on a full supply and run it cold than it is to employ the watchful regulation required to maintain a temperature approximating 85 deg. C. There are several reasons why the jackets should not be run too cold, even when there is an unlimited supply of water to be had without expense for pumping. In the first place, a jacket run at a temperature of say 30 deg. C. steals a great deal more heat from the interior of the furnace than one run at 85 deg. C. It is idle to waste heat in raising the temperature of water when it might be employed in smelting ore. Again, this same undue abstraction of heat from the furnace favors accretions and black tuyeres, with their attendant train of evils. I have seen a furnace, which demanded almost constant punching and barring to keep it alive, nearly cured by running the jacket-water so hot that it steamed.

Another scarcely less important reason for keeping the water moderately hot is to avoid the injury to the jackets which is likely to result from running them too cold. This is especially noticeable with cast-iron jackets, but is, I believe, almost as bad for our ordinary jackets, though not so immediately striking in its results. Finally, all of the various kinds of metal of which jackets are constructed do not develop their best qualities of strength and elasticity until a temperature is reached far above the boiling point of water.

The device of cooling the jackets by the actual vaporization of the water has been developed by one of the leading manufacturers of smelting furnaces until it has become an established system. I have no personal experience in the employment of this method, but it is based upon thoroughly scientific principles, and statements from a considerable number of metallurgists who

have used it are favorable. It seems, therefore, a subject worthy of serious investigation for plants which suffer from a scarcity of water.

This system of cooling is, of course, based upon the extraordinary absorption of heat resulting from passing from the state of liquid to that of vapor. If, for instance, cooling water is supplied to the jacket at 17 deg. C., and is discharged at the boiling point (100 deg. C.), it absorbs a certain amount of heat which we may call "x"; but if this same quantity of feed-water at 17 deg. C. be vaporized to steam, it will have absorbed about 7.5 x. The jacket is run no hotter in the one case than in the other, and we have used only one-seventh as much cooling water.

The serious corrosion of jackets is not a common occurrence; still, it is a grave matter when it does occur, and the record of such cases is too long to pass over without comment.

We may dismiss, to begin with, that class of troubles which arises manifestly from the presence of acids or sulphates in the cooling water. Indeed, we may go farther, and eliminate all cases of jacket-corrosion in which the steam boilers, using the same kind of water, suffer no especial corrosion or inconvenience. I am not underestimating the difficulties, delays, and dangers that result from the presence of the impurities to which I have just referred. I eliminate them simply because they are familiar to every one; and, being familiar, their treatment would be superfluous in this connection. As a rule, the treatment that is good for the boiler is good for the jacket: frequent blowing-off, removal of all scale and sediment, neutralization — or other chemical treatment — of the feed-water, and similar familiar, common-sense remedies.

The cases of corrosion to which I refer are much more subtle and mysterious, and produce grave injury from apparently insufficient cause.[1]

The corrosion of the jackets at the Copper Queen smelter at Douglas, Arizona, is accompanied by so many contradictory features that it cannot be ascribed solely to the composition of the

[1] Considerable information upon this subject will be found in a paper by George E. Lee, Supt. Copper Queen Consolidated Co., of Arizona, entitled "The Corrosion of Water-Jackets of Copper Blast-Furnaces," *Trans. A.I.M.E.*, Vol. XXXVIII, p. 877.

cooling water. This is derived from wells 600 ft. in depth, and is also used for the steam boilers. Mr. Lee gives the following analysis of the solids carried by this water: [1]

	Grains per U. S. gallon.
silica	0.861
iron oxide and alumina	0.223
calcium carbonate	0.261
calcium sulphate	—
magnesium carbonate	—
sodium and potassium sulphates	14.850
sodium and potassium chlorides	9.732
sodium and potassium carbonates	6.482
	32.409

Its most noticeable feature is its comparatively high content of sodium and potassium sulphates, but this does not seem capable of effecting the serious injury which accompanies its use.

Mr. Lee states that the inner plates of the jacket, which are of the ordinary high-grade steel commonly used for the purpose, are one-half inch (0.012 m.) thick, and that in from 10 to 12 months they have been reduced by corrosion to a thickness of from $\frac{1}{8}$ to $\frac{1}{16}$ inch, while the outer plates are eaten about one-half as much. The pitting, or corrosion, is irregular. The same trouble is experienced in some wrought-iron pipe lines where the water is not hot. The cast-iron impellers of the rotary pumps which circulate the water for the condensers in the power house are badly corroded, being eaten, in certain local spots, to a depth of $\frac{1}{4}$ inch. Strange to say, the eight 500-h.p. Sterling boilers using the same water are apparently not attacked.

Mr. Lee sums up the matter as follows: "It will be seen that materials as different as flange-steel and cast-iron are both attacked by the water; that steel exposed to very hot water, such as exists in the boilers, is apparently not attacked; that pipes handling water not over 24 deg. C. are attacked; that pumps handling water both cold and moderately heated are attacked; and that steel surfaces heated on one side, and with water on the opposite side of, say, 60 deg. C. are badly attacked, while plates cooled on the one side by the air, and exposed to the same water, are very little attacked."

[1] One grain = 0.0648 grammes
One gallon = 3.785 litres

The matter has been studied extensively at this smelter, the following causes of the trouble, amongst others, being considered: air bubbles lodging on the iron, from carbon dioxide in the water; electrolytic action between the iron and its oxides, or similar causes; chemical action. Nothing definite has been determined, but the management informs me in a recent letter that the addition of a small amount of heavy oil to the water going to the jackets has diminished the evil somewhat, probably by mechanical protective action.

Hixon, in Ontario, Canada, has had similar difficulty while using the water from the peat bogs of that country. It did not affect the boilers, but damaged the fire sheets of the furnaces so seriously that copper sheets were used (successfully) in their stead.

At Mount Lyell, Tasmania, this same trouble prevails in an aggravated form. The original steel jackets of the pyrite furnaces were destroyed so rapidly that recourse was had to cast-iron jackets, such as were formerly so universally employed in lead smelting. This lessened the evil decidedly, but of late years a further improvement has been made by abolishing the water-space in certain of the jackets, and substituting a pipe-cooled solid jacket of cast-iron.

It will be noted, therefore, that our remedies for mitigating the corrosion of water-jackets by cooling water which is not manifestly unfit for the purpose (such as acid waters from mines) seems to be limited to four alternative measures. (*a*) Using a little heavy oil in the feed-water to serve as a protective coating. (*b*) Employing copper for the inner sheets of the jacket. (*c*) Using jackets of cast-iron, with the ordinary water-space. (*d*) Using pipe-cooled cast-iron jackets without any water-space.

The whole subject still remains in an unsatisfactory condition.

The following figures give some idea of the amount of water that may be required to cool the jackets under ordinary conditions. The proportion of water used at Cananea seems high, but this is accounted for, in part, by the fact that it is discharged at an unusually low temperature — about 40 deg. Centigrade.

JACKET-WATER USED BY BLAST-FURNACES

Plant	Base	Size of Furnace in Inches	Height of Jackets	Area of Jacketed Surface in Square Feet	Gallons of Jacket Water per Minute	Gal. per Minute per Square Foot Jacket Area
Cananea.....	Cu	48 x 210	150¾″	552.	340	16
Omaha and Grant.	Pb	36 x 144	40½″	105.	40	8
Granby[1].....	Cu	44 x 266½	183″	806.	250	.31

The importance of having a large surplus tank-capacity at a sufficient height above the furnace to feed the jackets by gravity is obvious, as it renders it temporarily independent of the pumping system. If serious disorganization of the feed-pipes should occur, a furnace-jacket will require only an exceedingly small supply of cooling water, if the blast is taken off, and the tuyere and breast closed so as to check combustion. If these measures are promptly and effectually executed, a large furnace — in normal condition — may be kept supplied with cooling water by hand until better relief is obtained.

The water pipes should be of ample size, and each jacket should be provided with two branch pipes, which should have flexible connections to prevent damage to the joints through the expansion and contraction of the jackets. The discharge water may be received in sheet-steel open gutters, and conveyed to tight pipes below the level of the tapping floor.

The portion of the furnace above the water-jacketed shaft consists, in the first place, of the framework for supporting the brick shaft and furnace superstructure. Its details may vary greatly, according to the designs of the different manufacturers; but it consists, essentially, of a strong framework of I-beams and channels, which is bolted securely to the main corner-columns, and which is entirely independent of all the water-jackets or lower portions of the furnace. Under certain conditions it is found advisable to construct the main girders of this frame in the shape of water-jackets.

The fire-brick wall which constitutes the upper portion of the

[1] Discharge from upper tier of jackets is feed for lower.

furnace-shaft, and which is continued upward to form the charging chamber, rests upon this mantel-frame and is held together by a strong system of ironing. The space between the upper edge of the upper jackets and the lower portion of the brick shaft is protected by sloping, ribbed, cast-iron plates.

Above the level of the charging-floor are the doors through which the furnace is fed. These are of various types, and are usually counterbalanced. They are often raised and lowered by pneumatic or hydraulic power.

The manner of connecting the furnace with the stack is subject to many variations, all of which have their advantages and their drawbacks. Perhaps the commonest plan is to continue the brick mantel (sometimes made of sheet-steel with air space for preheating the blast) up to a sufficient height above the charging-floor, and then connect it with the dust chambers, or large flue, by means of one or more sheet-steel, brick-lined downtakes, or goose-necks.

At other times the charging-floor is constructed considerably higher above the furnace than is customary in the preceding plan, and the downtake for the smoke comes off from the brick shaft below the charging-floor. This obviates the necessity of any superstructure above the charging-floor except, possibly, a light (sometimes movable) hood and stack for use when blowing in the furnace.

Whatever plan is used, it is important that the construction of the goose-neck or downtake which connects the furnace and dust chambers be so designed and executed that its lower extremity shall rest against a firm, independent support, and that it shall not be permitted to deliver its powerful thrust against the walls or roof of the dust chamber. Also, that the whole superstructure and downtake shall be built in such a manner that the men who charge the furnace shall have ample opportunity to move and breathe, and shall not suffer unnecessarily from radiation of heat.

Moreover, it is of importance that the downtake shall be of ample dimensions to fulfil its purpose. Knowing that the draft of the blast-furnace is obtained by pressure of wind from below, and not by suction from above, and apparently neglecting all consideration of the great checking of the draft which results when the charging-doors are opened, some designers cut down

the area of their waste-flues far below suitable limits. The result is a two-fold train of evils, both of which are composite and far-reaching.

In the first place, the escape of the heat and furnace gases renders the work of charging so unendurable that the men either cannot, or will not, execute it properly. This is certain to cause improper running of the furnace, and consequent irregularities in succeeding operations. Again, an insufficient draft prevents the free escape of the waste gases, and encourages over-fire — or mounting of the heat to the top of the charge. I have seen this unpleasant — and almost always costly — condition relieved, in several instances, by improving the connections between furnace and stack so that a proper draft could be obtained.

The method pursued in charging the furnace also varies considerably at different plants. While certain metallurgists, who are managing furnaces of moderate size, still adhere to the old hand method of charging and insist that their furnaces run enough better to pay the extra cost, it is manifestly impossible to adopt so slow and expensive a plan for the larger type of furnace, and it seems to be only a question of time when this form of labor shall become extinct.

The largest furnaces — which stand usually with their long axis parallel to the long axis of the building — have a line of rails running along either side, and charging-doors spaced in such a manner as to correspond to a line of side-dumping railway cars of especial construction, which bring the ore alongside. Each car is filled at the ore-bins with its proper quota of bedded ore, first-class ore, limestone, etc., so that each of these loads represents independently perhaps 5 tons of properly fluxed smelting-mixture. The charging-doors of the furnace being raised by compressed air, the car-bodies are tilted by the aid of pneumatic pistons which swing from above, and the charge slides quietly into the furnace. The next train comes in on the other side, and thus a reasonably good distribution is secured.

In cases where the gases are taken off below the charging-floor, bottom-dump cars may be run directly over the open top of the furnace. In such cases it is found that the impact of the falling mass consolidates the contents of the shaft to an undesirable extent; this evil may be obviated by introducing two or three rows of steel rails which, being staggered, receive the main

force of the falling ore, and permit it to drop gently into the furnace.

Smaller furnaces are charged by means of steel charging-barrows, or by other methods which will be considered later.

The particular influences which the metallurgist desires to exert in charging his furnace are so variable that the subject does not lend itself to general treatment, but may be studied to the best advantage in connection with specific cases.

An interesting method of charging furnaces direct from bins was installed by Robinson, in 1909, at the Teziutlan smelter in Mexico, and described in detail by him.[1]

Above each of the Teziutlan furnaces there is a 65-ton (59 m. t.) bin, the charging-cars running directly over the top of this row of bins. These bins contain four dividing-plates, extending half way up, and thus cutting the bin practically into five sections. Each of these compartments has two gates, each gate being operated from a hydraulic cylinder. All of the 10 gates are controlled from one bank of levers in front of the furnace and, by looking into the furnace, it can be seen at what point it is desirable to feed a charge. There are, therefore, 10 separate points at which the furnace can be fed from this bin, which ensures a distribution of the charge about as satisfactory as can be obtained by hand-feeding.

The efficiency of charge-handling is improved materially, as the delivery of the charge to the furnace bins can go on — up to the limit of their capacity — independently of the feeding of the furnace, and the furnace can be kept full to the top mantel-plate, thus maintaining a far greater uniformity of conditions than is practicable with the customary intermittent feeding.

Many varieties of charging-cars are in use, a late type of which is shown in the accompanying illustration.[2]

This car is in use for charging the blast-furnaces at the Granby smelter, British Columbia, and is designed especially to secure an accurate distribution of the fuel and ore, which are charged separately. Each car is divided into four compartments, and is supplied with two sets of wheels; the lower, for running on the ordinary track, the upper near the top, for supporting the car on the rails set in the side walls of the furnace when the little

[1] *Eng. and Mining Journal*, Oct. 2, 1909.

[2] By permission of *Engineering and Mining Journal.*

train of three cars enters the furnace. When they are completely inside, the arm A is pulled by means of a long iron hook, the locks are released, and the contents fall upon the charge below. In actual working, the time elapsing from the moment the train enters the furnace until it again emerges varies from 10 to 20 seconds.

The sequence of operations for one charge is as follows: the train of three cars, drawn by a 30-horse-power electric locomotive,

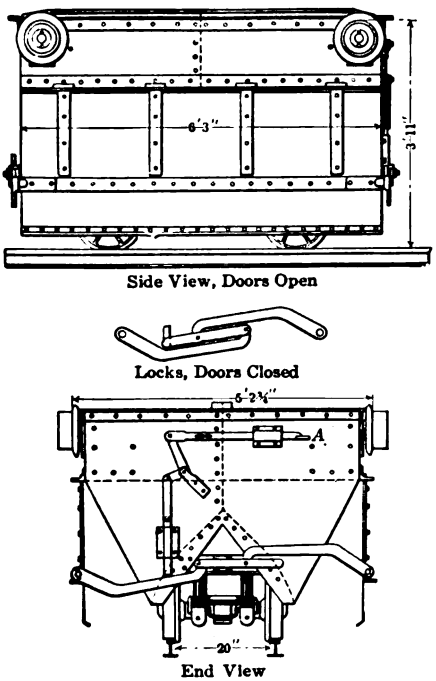


FIG. 47. — Furnace-charging car

is run under the fuel bins to receive a charge of 1¼ tons of coke, which is dropped into the furnace as described. The cars are then filled with 10 tons of ore-mixture which is run into the furnace and dumped; with the withdrawal of the cars from the furnace, the cycle is completed.

The division of the car into four compartments, and the impulse of the sliding ore toward the sides of the furnace, prevent definite separation of coarse and fine particles. What little

segregation occurs is that of the coarse ore toward the center, and at the very outside, the finer portion occupying the intermediate space. This is the desired arrangement, as it ensures the most uniform smelting under the conditions at Granby. Of course it is simple to omit the ore or coke charge from one or more compartments, or to vary it at will in harmony with the requirements of the furnace.

The blast-system of the furnace consists of the blower, pipes, and connections, but does not necessarily include the power by which the blowing machinery is driven.

Ordinary fan blowers, while low in first cost, delightful to run, and capable of delivering a large volume of air at low cost, are not regarded as suitable for the pressures now commonly employed in copper smelting. The strongest collateral evidence for the correctness of this position is that the great smelting companies do not find it profitable to use them, and it is quite certain that such would not be the case if it could be demonstrated that the fan blower was an advantageous machine for their purpose.

There remain, therefore, for the consideration of the ordinary copper metallurgist, two great types of blower, to which may be added a third one that possesses certain interesting features, and that has been used sufficiently to entitle it to consideration. These are: blowing-engines, rotary-piston blowers, and turbo-blowers.

It is not the purpose of this book to describe the mechanical details of standard apparatus, and this reservation applies particularly to the subject of blowers. Even if the theoretical side of the flow of air in pipes, and of certain other questions pertaining to the compression, transmission, and expansion of air, are not in an entirely satisfactory condition, the metallurgist will find slight cause for complaint of the manner in which these matters are applied in practice. Of all departments of the smelting plant, I know none which is so easily managed, so satisfactory, makes so little trouble, and comes so near doing what was anticipated of it — provided always that the blowing plant is selected judiciously, erected properly, and run well within its capacity.

Such small space as can be spared to this subject will be devoted to a brief consideration of the principles involved in connection with the points just enumerated.

While the blowing-engine, depending upon the reciprocating motion of a tightly enclosed piston, is the typical apparatus for the compression of air beyond a certain limit, it is not designed for delivering the large volume of air at low pressure which is demanded by the copper blast-furnace. Below a pressure of six or eight pounds per square inch there are certain losses which become so serious as to diminish greatly the economy of the result.

In the gap, where the pressure is too great for the fan blower, and still too low for the advantageous working of the piston-blower, the rotary-piston blower has its place, and is the usual type employed at copper blast-furnaces.

Some years ago when the increase in size and height of furnaces, and in the rapidity of smelting, became so rapid, many rotary-piston blowers were called upon for work of a nature for which they were not designed, and which they were sometimes unable to perform. The manufacturers, however, rose to the situation, and now furnish rotary-piston blowers of a size unknown at that time, and which yield, with efficiency and economy, pressures beyond anything called for in copper smelting.

The operation of the rotary-piston blower consists in revolving, in an enclosed case, two curved impellers, the surfaces of which during rotation remain in a definite relation to each other and, consequently, while almost in contact maintain always a definite and uniform clearance. The air thus enclosed between these impellers and the surface of the case is discharged through the blast-pipe in a practically continuous stream, as a certain fixed volume of air is drawn in and forced out twice during each revolution of an impeller — or four times during the revolution of the two impellers.

It will thus be seen that, providing the air that has once been enclosed between the impellers and the case does not leak back, the rotary-piston blower is a positive displacement machine, and that the power required to operate it should stand in definite proportion to the amount of air delivered. As a matter of fact, this leakage has been reduced to a comparatively small amount and, with the modern large blowers of the best makes, pressures up to at least eight pounds per sq. in. are guaranteed, with high efficiency, and a correspondingly low slip and leakage.

The absence of valves, and the reduction of all moving parts to two rotating units, render the machine peculiarly simple and

capable of withstanding rough usage, but do not obviate the necessity of reasonable care, and especially of keeping the impeller shafts in accurate alignment.

As an evidence of the remarkable increase in capacity which this machine has undergone during the past few years, I may mention that the largest rotary-piston blower of only 15 years ago displaced about 70 cu. ft. (2 cu. met.) air per revolution, while the newer Connersville blowers at the Washoe smelter displace 320 cu. ft. (9.14 cu. met.) air per minute, have journals 10 inches in diameter, and weigh 130,000 lb. (59 met. tons). The pressure is 40 oz. per sq. inch.

On the whole, the rotary-piston blower seems still to be re-

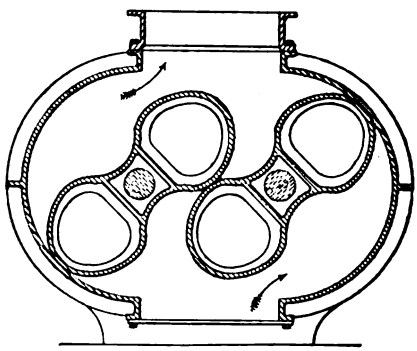


FIG. 48. — Cross-section through center of rotary-piston blower

garded as the most advantageous type of blowing apparatus for the copper smelter.

In making this statement, I do not wish to be understood as slighting the claims of the turbo-blower. I am simply giving it brief notice because we have not, in this country, sufficient positive information of its application to copper blast-furnaces to enable us to pass judgment upon its merits. The most useful and reliable statements that we possess regarding this apparatus come from Robert Sticht, of Mount Lyell, Tasmania, where the Parsons turbo-blower is used on a large scale and with satisfactory results. The following information regarding their use at the Mount Lyell pyrite smelter is derived from Sticht's description of the Mount Lyell mine and smelter in Vol. XVI of *The Mineral Industry*, and from personal letters received from him; also from

a general statement from George W. Wright, chief engineer at Mount Lyell.

The rotary-piston blowers at the Mount Lyell smelter were wearing out, and a decision to raise the height of the blast-furnaces and to increase the blast pressure led to the installation of a Parsons steam-turbine-driven turbo-blower, capable of supplying 18,000 cu. ft. of free air per minute, of 54 oz. pressure, 460 brake horse-power.

This proving satisfactory, two further Parsons turbo-blowers of the same nature were installed, each of 1025 brake horse-power, and delivering 36,000 cu. ft. free air per minute at a 64-oz. pressure. The steam consumption of the turbo-blowers is found to be 13½ lb. per brake horse-power. There is ample water at Mount Lyell for condensing purposes.

After trial for a considerable period, I am informed by Sticht that the apparatus is eminently satisfactory, and that its efficiency continues with few repairs.

At Rio Tinto, Spain, the same type of turbo-blowers as those at Mount Lyell are used on six pyrite furnaces, and are stated by the management to yield satisfactory results.

CHAPTER VII

BLAST-FURNACE SMELTING

THE first question which confronts the metallurgical author, when attempting to write upon the familiar subject that heads this chapter, is to know what definition to attach to it.

The blast-furnace itself, viewed as a piece of metallurgical apparatus, presents no unusual complications; but, as soon as we come to consider the processes which are executed in it, we are forced to ask whether — when using the term "blast-furnace smelting" — we mean the melting down of oxidized ores in the blast-furnace, or the bessemerizing of pyrite ores in the blast-furnace, or the peculiar half-oxidizing, half-reducing method whereby we overcome the reducing influence of coke by using an excess of blast, and oxidize the sulphur and iron to reinforce our scanty coke charge. In other words, do we mean ordinary melting-down smelting, or true pyrite smelting, or partial pyrite smelting?

It is idle to allow ourselves to be misled by the fact that we apply the same name to the apparatus in which all these different processes are executed. Though the furnace may remain the same, the methods themselves are as different as can well be imagined; and though the end-results attained (slagging of the iron and silica, and production of a satisfactory copper matte, are identical, the means employed for attaining these results are so completely divergent that any attempt to treat them under one head can only result in confusion.

All authors have recognized the justice of this statement so far as regards the absolute difference between ordinary coke-smelting and the other two processes which I have just cited, and there has never been any ambiguity in this connection; but when it comes to the discussion of pyrite smelting, there is nearly always an attempt to make this overworked term include the third, and quite distinct, operation which commonly goes by the name of "partial pyrite smelting" or "semi-pyrite smelting." [1]

[1] I object to the term "pyritic smelting" because it was long ago employed by Percy to designate a totally different operation: namely, the simple fusing,

So far from being so nearly allied as to be included under one head, I regard "pyrite smelting" and "partial pyrite smelting" as processes so divergent, and yet so liable to confusion from the fact that a little coke is used in true "pyrite smelting," that writers should take especial pains to maintain their identity under all circumstances; and, as no clear definition of partial pyrite smelting had been enunciated, I suggested in "Principles of Copper Smelting," page 323, that we should draw the line between true and partial pyrite smelting at that point where so much coke is used that its influence upon the oxidizing power of the focus begins to become plainly discernible. The justice of this definition may be better appreciated after one has made himself thoroughly familiar with the reactions that occur within the furnace-shaft when smelting raw sulphide ores in an oxidizing atmosphere.

Omitting the comparatively unimportant instances in which natural oxide or carbonate ores are smelted for the production of metallic copper, it is difficult to find to-day, in this country, a good illustration of the pure reducing-smelting process as applied to roasted pyritic ores of copper. Only a few years ago the blast-furnace was our typical reducing apparatus, and, with the exception of a little work by Long at Ely, Vermont, and by Bartlett at Portland, Maine, it was confined almost entirely to the treatment of ores which had already had their sulphur and iron content oxidized by the preliminary operation of roasting.

The reverberatory was regarded distinctly as our especial oxidizing furnace because — although it showed little active influence in that direction in itself — it did not at least interfere with those reactions of oxides upon sulphides which play their useful part by virtue of their fixed oxygen,[1] and did not appropriate this dearly-bought fixed oxygen to its own purposes as did the reducing blast-furnace. These reactions in the reverberatory result, usually, in removing some 20 to 25 per cent of the sulphur still remaining in the partially roasted ore, and increase the capacity of the roasting plant correspondingly.

At present, however, the reverberatory smelting furnace is regarded as but a feeble apparatus for the removal of sulphur, while the blast-furnace is run on raw sulphide ore in such a manner

with carbonaceous fuel, of pyritic ores without any attempt at oxidation of their constituents.

[1] For instance: $Cu_2S + 2CuO = 4Cu + SO_2$.

that from 70 to 90 per cent of the sulphur is driven off and its iron oxidized and slagged.[1]

It has been found that a great saving in coke could be effected by utilizing, in the blast-furnace, the heat evolved by the oxidation of the sulphides; and this advantage alone — apart from the saving of roasting-costs, losses, and plant — would be sufficient in most cases to constrain the metallurgist to burn and smelt his sulphide ores in one and the same furnace. Consequently, at the present day, the blast-furnace has come to be the great oxidizer, while the reverberatory has been relegated to the more modest rôle of a melter-down of inert, oxidized material.

In the lack of suitable smelting plants which might be employed as illustrations of the purely reducing, melting-down type of blast-furnace processes, I shall defer all discussion of the details of furnace management to the section on partial pyrite smelting, as it is in this class that we shall find most of the great copper blast-furnaces of the present day, especially in the United States.

Regarding for the moment the blast-furnace in its original capacity as a simple melter-down of already oxidized ores in a reducing atmosphere, I will point out its distinctive reactions, and illustrate its operations by simple examples from practice.[2]

The following general statements pertain to the reactions occurring in the ordinary coke-burning blast-furnace process, in which roasted sulphide copper ores are smelted without attempt at oxidation.

A proper furnace-mixture will contain such proportions of silica and bases as will, on melting, yield a suitable slag (see p. 9). If the ores at the disposal of the metallurgist will not produce such a mixture, the lacking constituents must be added.

[1] At this point I desire to urge metallurgists not to regard this oxidation of sulphur in the blast-furnace entirely as a roasting of the ore. While the final result achieved is identical with that obtained from the former combined operations of roasting and smelting, the oxidation of the sulphides in the true pyrite furnace in no wise resembles these placid operations. On the contrary, it is largely a bessemerizing operation, and should be regarded as such to avoid misconception.

[2] In cases where this book is used by beginners, this section should be studied in connection with Chapter IV (the behavior of ores, etc.). Those desiring to pursue the subject more thoroughly should consult chapters IV and VI in "Principles of Copper Smelting." The brief treatment of furnace reactions in this chapter must be regarded only as a general, comprehensive view of the matter, and not as laying claim to scientific accuracy.

The mixture must also be of such a nature that it will yield a matte of a suitable richness in copper (incidentally, perhaps, gold and silver).

For reasons which will soon be obvious, it is convenient to begin the consideration of such a mixture with the study of the matte rather than the slag. Indeed, a glance at a preceding paragraph will explain this necessity. It is stated there that the slag will consist of the silica and bases contained in the ore-mixture. Analysis will furnish us with the exact amount of silica present, as well as with the iron, lime, magnesia, alumina, and other rarer oxides — all of which are termed bases.

Having this accurate analysis, it would, then, be quite simple to calculate the composition of our prospective slag, provided that all of these bases remained as bases (oxides), and entered the slag. This is, however, not the case, as one of our most important bases — iron oxide — is, in part, reduced to Fe and, combining with sulphur, enters the matte as FeS. Thus a portion of our iron is lost to us as a base, and must be figured as a part of the matte.

We cannot, therefore, tell how much iron will be available as a base for the slag until we have determined how much of it is going into the matte as FeS, and this is the reason why it is necessary to begin the study of the composition of the slag by determining first how much iron will go into the matte.

Matte consists, essentially, of the sulphides of the ore-mixture after they have undergone such modifications as they suffer during the process of fusion.[1]

The following rules will be found useful in familiarizing one's self with the mutual behavior of the three important matte-forming constituents. They are not entirely accurate, but are subject to corrections which may easily be applied when the subject has become familiar; but as a foundation for beginners they are invaluable.

The three rules of matte-formation in a reducing atmosphere:

First Rule. — The amount of sulphur left in the roasted ore determines the amount of matte that will be produced in smelting that ore.

Second Rule. — The copper in the roasted ore has the first call

[1] See "Principles of Copper Smelting," Chapter X, for a detailed study of matte.

on the sulphur. Each pound of copper takes up about one-quarter of a pound of sulphur, forming cuprous sulphide (Cu_2S).

Third Rule. — After all of the copper present has taken up the sulphur that it requires, according to Rule No. 2, the remaining sulphur takes up iron at the rate of one and three-quarters pounds of iron for each pound of sulphur, forming ferrous sulphide (FeS).[1]

Illustration of the application of the three rules of matte-formation:

A furnace-charge composed of roasted sulphide ores is to be smelted in the blast-furnace. A partial analysis of this charge shows that it contains copper, 7 per cent; sulphur, 6 per cent; iron, 42 per cent. What will be the weight of the resulting matte, and what per cent of copper will this matte contain?

Calculation (without certain corrections which must be applied in practice).

It is always easiest to base calculations of this nature upon 100 pounds (or parts) of the original mixture, because then pounds and per cents will be identical.

Application of the first rule:

Under this rule we desire only to learn the amount of sulphur in the ore under consideration. This we are told already in the problem: namely, that 100 lb. of the ore contains 6 lb. sulphur.

Application of the second rule:

One hundred pounds of the ore contains 7 lb. copper. One pound copper unites with one-quarter pound sulphur to form Cu_2S; consequently,

7 lb. copper unite with 7×0.25 lb. = 1.75 lb. sulphur to form $7 + 1.75 = 8.75$ lb. Cu_2S

Application of the third rule:

After the copper is satisfied, the remaining sulphur takes up 1.75 lb. iron per pound sulphur.

We had originally	6	lb. sulphur
the copper has used up	1.75	" "
we have remaining	4.25	" "
1 lb. sulphur takes up	1.75	lb. iron
4.25 lb. sulphur takes up 1.75×4.25 =	7.44	" "

forming $4.25 + 7.44 = 11.69$ lb. FeS

[1] See "Principles of Copper Smelting," pp. 60–68, for demonstration of these rules.

Our matte-compelling constituent — sulphur — is now entirely provided for, and the two sulphides which it has formed mix together as matte.

We have Cu_2S	8.75	pounds
FeS	11.69	"
Total matte	20.44	"

As this 20.44 lb. of matte (under our present assumption) consists solely of copper, sulphur, and iron; and as we know that it contains exactly 7 lb. copper, we may express its copper content by the fraction $\frac{7}{20.44}$ which (reduced to decimals) = 0.3425 — or 34.25 per cent copper.

Thus the weight of the matte produced from 100 lb. of the ore-mixture is 20.44 lb., and its assay in copper is 34.25 per cent.

A more generally convenient way of determining the amount of copper (or of gold or silver) in matte is by multiplying the assay of the original ore by the ratio of concentration: for instance,

20.44 lb. matte was produced from 100 lb. of the original ore, then 1 lb. matte was produced from $\frac{100}{20.44} = 4.89$ lb. of the ore; or, reversing the statement, 4.89 lb. ore produced 1 lb. matte. Thus the ratio of concentration is 4.89 to 1, and the matte must be 4.89 times as rich in copper as the ore.

The ore assayed 7 per cent copper; then the matte must assay $7 \times 4.89 = 34.23$ per cent copper, which proves the preceding result.

To show the advantage derived from the employment of such a factor of concentration, I will extend this problem by supposing that the original ore which was the subject of the preceding calculation contained — in addition to its 7 per cent copper — 4.3 oz. silver and 0.08 oz. gold per ton. What would the matte assay in these metals?

Having already determined that 4.89 parts ore produce 1 part matte, we need only to multiply the assay of the ore in silver and gold by this factor of concentration.

original ore assayed	4.3	oz.	silver	per	ton
matte assays	$4.3 \times 4.89 = 21.0$	"	"	"	"
original ore assayed	0.08	oz.	gold	per	ton
matte assays	$0.08 \times 4.89 = 0.39$	"	"	"	"

Having, for the sake of clearness, explained the above problem

without applying the ordinary corrections that would be made in practice, I will now repeat the calculation, making deductions for metallurgical losses — which will vary at each smelter, and which can be determined with accuracy only by elaborate investigations extending over considerable periods of time.

We may assume that 5 per cent of the total copper, 6 per cent of the total silver, and 2 per cent of the total gold will be lost — in this particular instance — during the melting of the roasted ore into matte.

In other words, the matte will contain only 95 per cent of the copper, 94 per cent of the silver, and 98 per cent of the gold, and it would seem that we may easily modify our preceding figures to conform to these new requirements by simply multiplying each original assay by the corresponding percentage of its recovery. This plan, however, would fail to take into account the loss of sulphur which will occur during the smelting of the ore and which, naturally, must exert a profound influence upon the assay of the resulting matte in copper, silver, and gold for the simple reason that, according to the first law of matte-formation, it is the sulphur which determines the amount of matte that will be formed.

While the conditions in the reducing atmosphere of the coke-burning blast-furnace smelting roasted ores are not favorable for the removal of sulphur, there is still a considerable escape of that substance as SO_2 gas, due especially to the familiar reactions between oxides, sulphates, and sulphides which occur, to some extent, in spite of the great affinity for oxygen of the carbon monoxide.

We do not, however, wish to include this removal of sulphur among our "metallurgical losses," as this term is reserved for the distinctively valuable metals of the charge. The amount of sulphur removed from such an ore, and during such a fusion as we are considering, is not large. For the present illustration I will assume it to be 12 per cent of the original sulphur.

Instead of attempting to apply these numerous modifications to the results already obtained, it will be easier to reconstruct the entire problem.

Problem. — A roasted sulphide ore contains 7 per cent copper, 6 per cent sulphur, and abundant iron. It also assays 4.3 oz. silver and 0.08 oz. gold per ton. Twelve per cent of the sulphur content escapes during smelting. The metallurgical losses are 5 per cent of the copper, 6 per cent of the silver, and 2 per cent of

the gold. What will be the weight of the resulting matte, and what will be its assay in copper, silver, and gold?

In cases of this kind I find it convenient to apply the corrections for losses to the original assays of the roasted ore:

	Original Assay	Metallurgical Losses	Corrected Assay
Copper	7 per cent	5 p. c. of original assay	6.65 per cent
Sulphur	6 per cent	12 p. c. of original assay	5.28 per cent
Silver	4.3 oz. per t.	6 p. c. of original assay	4.042 oz. per ton
Gold	0.08 oz. per t.	2 p. c. of original assay	0.0784 oz. per ton

Adopting these modified assays, we begin our calculation with an ore containing copper, 6.65 per cent; sulphur, 5.28 per cent; silver, 4.042 oz. per ton; gold, 0.0784 oz. per ton.

Following out the necessary steps — as in the preceding example — we find that we shall have

$$\begin{aligned} Cu_2S &= 6.65 \times \tfrac{5}{4} = 8.31 \text{ lb.}^1 \\ FeS &= 3.62 \times 2.75 = 9.96 \text{ "} \\ \text{Total matte,} &\quad 18.27 \text{ "} \end{aligned}$$

As this 18.27 lb. matte has resulted from the smelting of 100 lb. ore, the ratio of concentration is $\frac{100}{18.27} = 5.473$ to one.

Multiplying the corrected assay values of the metals in the original ore by the factor of concentration, we have

	Corrected Assay of Ores	Factor of Concentration	Assay of Matte[2]
Copper	6.65 p.c. ×	5.473 =	36.4 per cent
Silver	4.042 oz. ×	5.473 =	22.12 oz. per ton
Gold	0.0784 oz. ×	5.473 =	0.429 oz. per ton

[1] The fraction $\frac{5}{4}$ is a factor which may be always used to convert any given weight of Cu into its weight as Cu_2S. In other words, 1 lb. Cu will produce 1.25 lb. CuS.

The fraction 2.75 may be used as a factor by which to multiply a given weight of sulphur to find what weight of FeS it will form. According to the third rule of matte-formation, "after all of the copper present has taken up the sulphur which it requires according to the second law, the remaining sulphur takes up iron at the rate of 1.75 lb. of iron to each pound of sulphur, forming FeS." Therefore, the ferrous sulphide which one pound of sulphur will produce consists of the 1 lb. sulphur plus the 1.75 lb. iron which it takes up — making 2.75 lb. of ferrous sulphide.

[2] Note that this simple conception of matte is employed for purposes

In studying the effect of the reducing blast-furnace operation upon roasted sulphide ores, we make frequent use of the process of elimination in order to divest our problems of all unnecessary material and make it easy to concentrate our undivided attention upon the main issue.

The calculation of ore-mixtures for the blast-furnace is usually made unnecessarily obscure by attempting to do too many things at once. The smelting of such a charge as I am now speaking of brings nothing new into our products except a certain amount of oxygen from the blast. The charge — in its broadest significance — consists of nothing except ore and coke.[1] To these is added nothing but some oxygen from the blast.

The products of the smelting operation, therefore, can contain nothing that was not already present in the ore and coke (plus the oxygen already indicated). It is plain, then, that the process of submitting these substances to a high temperature causes simply a rearrangement (due to the awakening of different chemical affinities by heat) and a change in form (due to the purely physical alteration which is also produced by heat).

A clear classification of results is a long step toward the comprehension of apparently complicated processes of this nature, and I purpose to classify the results of this blast-furnace operation with a view to simplifying its study by the elimination of one after another of the products (after submitting each to such examination as its importance demands), until we arrive at a single remaining substance which is alone worthy our consideration (in this particular connection). I intend to go still further, and hope to show that

of illustration, and falls far short of its actual composition, though most useful as a foundation for study.

The most essential points to recollect are: first, that in actual mattes the amount of sulphur is almost always too little to satisfy the theoretical requirements of their copper and iron content; second, that as Cu_2S and FeS are the lowest possible combinations with sulphur, it is plain that one or the other of these metals can only partly satisfy its craving for sulphur; and that, as copper has the greater affinity for sulphur (see first rule of matte-formation), we arrive — by process of elimination — at the conclusion that the excess metal which is present in the matte must be iron in the metallic form; and this is borne out by our study of the behavior of FeS at very high temperatures (page 45), as well as by minute analytical and metallographical studies of mattes. See chapter on "Matte" in "Principles of Copper Smelting."

[1] For the moment, I will include all flux, foul slag, etc., under the head of "ore."

even this sole remaining object of study — the slag — may, by simple cancellation of its constituents, be brought to so tangible a form that, for commercial purposes, it may be represented by a few tons of excess silica, or excess iron, or excess lime, and may thus be presented in a manner which shall not only be entirely within the comprehension of the non-technical reader, but shall also be scientifically correct so far as it pretends to go.

All of the products resulting from the smelting of roasted sulphide ores in the coke-burning blast-furnace may be classified under two heads, each head having two divisions.[1]

I. Products neglected by the metallurgist in the calculation of his furnace-mixture.

(*a*) gases (*b*) flue-dust

II. Products upon which the attention of the metallurgist is concentrated in the calculation of his furnace-mixture.

(*a*) matte (*b*) slag

I. *Products neglected by the metallurgist in the calculation of his furnace-mixture.*

The two products which form this first division are of no importance to the metallurgist in calculating his furnace-mixture, because they have no direct bearing upon the purpose which he has in view in undertaking the operation of smelting ore. This purpose is two-fold: first, to produce a slag which shall, at a suitable temperature, melt into a sufficiently liquid condition to flow out of the furnace and to permit the matte to settle out of it; second, to produce a matte of suitable composition, which shall contain the copper, gold, and silver that were in the ore-mixture. He would be very glad if he need only give his attention to the second of these aims but, unfortunately, he cannot attain the second condition without fulfilling the first one.

(*a*) *The gases.* — While the gases from the blast-furnace often demand careful consideration on account of their injurious effect upon surrounding vegetation, they have no bearing upon the point in question. Whether they are noxious, or indifferent, or valuable, they have nothing to do with the calculation of the solids of the charge.

[1] I base this classification upon roasted sulphide ores rather than upon natural oxides because the former is the more comprehensive situation and, in simpler form, applies equally well to the latter.

They consist of products of combustion, of water vapor, of the nitrogen that belonged to the air introduced by the blast, of SO_2 and SO_3, and of small amounts of all of the volatile constituents of the charge. They are intangible and escape at once from the process and from the attention of the metallurgist — so far as this especial problem is concerned.

(*b*) *The flue-dust.* — This consists mainly of fine ore-particles carried away from the furnace by the draft. It nearly always contains recoverable values, and its collection and subsequent treatment are matters of anxious consideration, but it has nothing to do with the question at issue.

We have now eliminated completely the first division of the products of the variety of smelting which we are considering, and have divested our ore-mixture of some of its difficulties, and of an important fraction of its weight.

The metallurgist, however, does not begin the calculation of his furnace-charge by any such elaborate and variable figuring as would be required to estimate the weight of the products forming this first division. By the use of rules allied to those which I shall formulate, he may say simply: "I find that the matte resulting from 100 lb. of my ore-mixture will weigh a certain amount; my slag will weigh a certain larger amount; all the rest will be products which I am not interested in at present." He thus obtains, by difference, the weight of his products which belong in No. 1 division.[1]

[1] As a matter of fact, in actual practice, the metallurgist does nothing of the kind. The probable weight of his matte he *does* figure out carefully as he desires to keep it up to a certain richness in copper; and the more weight of matte that he produces from a given ore, the poorer must that matte be in copper, because the increase in its weight can come only from an increase in its proportion of ferrous sulphide. But the weight of slag that he is going to produce does not interest him. He figures his furnace mixture so that each pound of silica present shall have its proper proportion of bases to form a slag of the composition which he desires, but he does not waste time in adding up the different substances to see what their total weight will be.

The withdrawal from the slag and matte of a certain proportion of the ore-mixture in the shape of flue-dust causes no complications worth mentioning because, although the withdrawal of this fine ore-dust affects slightly the total weight of the resulting slag and matte, it does not affect their composition to any noticeable extent.

II. *Products upon which the attention of the metallurgist is concentrated in the calculation of his furnace-mixture.*

We now come to the products of the smelting operation upon which the entire attention of the metallurgist is concentrated when he is calculating the composition of his furnace-charge.

(*a*) *Matte.* — This substance has already been studied sufficiently to give a reasonable idea of the manner in which the metallurgist would determine, in advance, its probable weight and composition.

After allowing for the proportion of the total sulphur which will be driven off from the roasted ore during its passage through the blast-furnace, and after establishing a corrected assay for the copper content of his mixture by deducting the probable metallurgical losses from the true assay, he simply follows the three rules of matte-formation, by which he distributes one-fourth of a pound of sulphur to each pound of copper, and then provides each pound of the remaining sulphur with 1.75 lb. of iron.

The sum of these three substances — copper, sulphur, and iron — may be taken as the total weight of the matte produced; and while this plan is based upon an incorrect assumption of the true composition of matte (which, incidentally, we do not know), it usually comes curiously near the truth and may be corrected, in each individual case and as soon as we have learned just how far it differs from the actual results, by multiplying it by some constant (established by trial) which shall convert the theoretical

The actual amount of elimination thus effected by throwing out the products belonging to the first division varies greatly according to local conditions, and is, for reasons already indicated, seldom known with any approach to accuracy. It is, therefore, particularly interesting when any one of the great plants goes to the pains of determining the proportionate weight of the furnace products for periods of time long enough to ensure reasonable accuracy. The following figures — fractions omitted — are from Ricketts' statements [1] of the weight of the products during the first six months of 1909, from a reverberatory furnace at Cananea running on roasted sulphide concentrates and flue-dust:

matte (and some foul slag)	25 per cent
clean slag	63 " "
volatilized (and flue-dust)	12 " "
	100

[1] *Opus cit.*

result into the actual weight obtained in average practice at that particular smelter.[1]

We have now disposed of three of the four products of the smelting operation and it will be useful, before the situation becomes complicated, to take stock of our material and see what we have remaining.

Whatever this remainder may be, it has all got to melt together into a liquid slag, and if, owing to lack of any one or more substances essential to its proper melting, it is unable to do so, these substances (flux) must be added before the mixture is charged into the furnace. Regarded in this manner, the calculation of a smelting-charge becomes an extremely simple matter, and is entirely within the capacity of any one familiar with simple arithmetic.[2]

I shall begin with an illustration of great simplicity, assuming that our ore-mixture consists solely of roasted iron and copper sulphides, with a predominating quartzose gangue, and that our available flux is a tolerably pure limestone. The smelting is to be conducted in a blast-furnace, and the fuel will consist of coke having an ash of such a composition that it will not be considered in calculating the slag that results from the smelting operation.[3]

The following analysis indicates the composition of the roasted

[1] In making matte calculations on mixtures containing the precious metals, it is customary to consider only the values of the latter, their actual weight being too trifling to include in the calculation.

An illustration of calculating the weight and value of the matte has been given in the preceding pages.

[2] In making this statement I am, of course, referring only to the simple process of calculating a slag of predetermined composition from the ores and fluxes at hand. The selection of the most fitting slag for any given situation, the determination of how far it is possible to go in abnormally increasing or decreasing any of its constituents, or the adaptation of a slag to peculiar local conditions, all demand a considerable knowledge of chemistry and metallurgy, and a still wider range of practical experience.

[3] I refer particularly to the ash of the coke because this is a solid mineral substance which, when the carbon of the coke burns away, remains mixed with the charge, and thus becomes part of the mixture to be smelted. It is a slag-forming substance, and as such demands consideration just as much as the ore itself, except that its quantity is very small.

In the present instance, I assume that the silica, alumina, iron, lime, etc., of which the ash is composed are present in proportions not far different from those of the normal slag which we intend to make. This fact, in connection with the extremely insignificant quantity of the coke-ash, will render

sulphide ore which is to be used to illustrate the calculation of a slag and matte under simple conditions.

The original composition of the raw sulphide ore from which this roasted ore is derived is of no interest to the metallurgist in this calculation. He is dealing solely with the roasted ore, and adopts its analysis as the basis of his calculations.

ANALYSIS OF THE ROASTED ORE

SiO_2	31.5 per cent
FeO	35.0 " "
MnO	2.0 " "
Al_2O_3	4.4 " "
CaO	3.0 " "
MgO	1.1 " "
Cu	7.0 " "
S	6.0 " "
Ag	4.3 oz. per ton
Au	0.08 oz. per ton
	90.0

There are several points about this analysis which are peculiar and which merit attention, as they are sure to be encountered in every analysis of roasted ore, and might as well be cleared up at once.

The most noticeable feature of the analysis, and one that would strike even the non-technical reader, is that its constituents do not add up to 100 per cent. Ten per cent is lacking, and it would be inconsistent to pass over such a glaring imperfection as this. It comes, of course, mainly from the oxygen content of the oxidized compounds referred to.

The next imperfection, or series of imperfections, although perhaps not obvious to the ordinary reader, would be peculiarly revolting to any one having even a rudimentary knowledge of metallurgical chemistry. His criticism would be that the analysis is given in terms which are impossible under the conditions to which the ore has been exposed (roasting); that it is utterly impossible that the iron of the roasted ore should exist as FeO, or the copper as Cu, or the sulphur as S; but that each of these substances would be distributed throughout the ore in a long series of peroxides, oxides, suboxides, sulphates, basic sulphates,

it unnecessary to pay any attention to it in our calculation; for while it will increase, by a few pounds, the total weight of the slag, it will have but an insignificant effect in changing its composition.

undecomposed sulphides, etc., and that it is misleading to express these substances in the simple form adopted by the assayer at the smelter.

This criticism would be quite sound if the desire of the metallurgist were to obtain a knowledge of the intimate composition of the roasted ore; but, as a matter of fact, he cares nothing at all about this point. Assuming that the ore has been properly roasted (which is a matter between the metallurgist and the roaster foreman), all that the metallurgist then desires to know is — how many pounds of iron and of copper and of sulphur, etc., are there in 100 lb. of the roasted ore?

He is not interested in knowing in what manner of complicated compounds this iron and copper and sulphur is tied up in the roasted ore, for the simple reason that the great heat in the blast-furnace will untie all of these bonds and redistribute all of these substances in such a fashion that a totally new and different set of compounds will result. Hence, the metallurgist has no concern with the unstable compounds which he is going to destroy during the next few hours, but cares only for the actual weight of the elementary (or, at least, stable) compounds with which he is going to form his new combinations in the blast-furnace; and I am now proceeding to point out what these new combinations will be. This point is so important, and yet so confusing, that it is worth while to consider the more important substances one by one.

It is the custom of the chemist to report each constituent of his analysis in the form which experience has shown to be the most convenient for the blast-furnace calculations.

SiO_2, for instance, is reported as such, because it is in this form that we figure it in our slag calculation.

Iron cannot exist as FeO, except in combination with silica or some other acid. It occurs in the roasted ore mainly as Fe_2O_3, Fe_3O_4, and undecomposed FeS. It is sometimes reported in the analysis simply as Fe. On the whole, however, it gives a clearer picture of its slag-forming power to report it as FeO, as this is the form in which it combines with silica to form slag.[1]

[1] The weight of the FeO may be expressed as Fe by multiplying by $\frac{7}{9}$. Hence, the 35 per cent FeO in the present analysis represents $35 \times \frac{7}{9} = 27.3$ per cent Fe. Conversely, the weight of metallic iron may be expressed as FeO by multiplying by $\frac{9}{7}$; $27.3\ Fe \times \frac{9}{7} = 35\ FeO$.

Manganese, having a combining weight almost identical with that of iron, and being almost identical with that metal in its slag-forming qualities, may be eliminated as a separate constituent (save in certain extremely rare instances) by simply adding it to the iron.

Alumina, lime, and magnesia, when present in such small quantities and with such a preponderating amount of FeO, may be added together and called "earthy bases." [1]

Sulphur, copper, silver, and gold have been already considered.

I present now a modified analysis of the roasted ore. This is merely the original analysis after it has been simplified in the manner just described, and with the deductions for metallurgical losses as explained on page 174.

ANALYSIS OF THE ROASTED ORE AFTER SIMPLIFICATION AND DEDUCTION OF METALLURGICAL LOSSES

SiO_2	31.5 per cent	
FeO	37.0	
Earths	8.5	
Cu	6.65	
S	5.28	
Ag		4.042 oz. per ton
Au		0.0784 oz. per ton

We will first determine what would be the composition of the slag derived from smelting the ore just as it is. In conformity with the preceding explanation, we must first establish the weight of matte that will be produced, in order to determine how much of the iron will be withdrawn from circulation.[2]

6.65 lb. Cu	$\times \frac{5}{4}$	= 8.31 lb.	Cu_2S
3.62 lb. S	$\times 2.75$	= 9.96 lb.	FeS
Total matte,		18.27 lb.	

As each pound of sulphur remaining after the copper has taken up such sulphur as it needs combines with 1.75 lb. iron (see third rule of matte-formation), the 3.62 lb. sulphur will require $3.62 \times 1.75 = 6.34$ lb. Fe, which is equivalent to $6.34 \times \frac{9}{7}$

[1] This last short-cut would not be justifiable at all if the mixture contained a sufficient proportion of these earthy oxides (especially, alumina) to exert a marked influence upon the quality of the slag. This point will be considered later.

[2] See page 171.

= 8.15 lb. FeO. Deducting this from the amount of FeO shown in the simplified analysis, we have 37 − 8.15 = 28.85 lb. FeO remaining for the slag.

The calculation is now completed, and we may construct a little table of the substances remaining in the ore-mixture to form slag:

TABLE OF SLAG-FORMING CONSTITUENTS IN 100 POUNDS OF ROASTED ORE

SiO_2	31.5 lb.
FeO	28.85 "
earths	8.5 "
Total	68.85 "

Thus 100 lb. of roasted ore will produce 68.85 lb. of slag, consisting of

SiO_2	$\frac{31.5}{68.85}$ =	45.75 per cent
FeO	$\frac{28.85}{68.85}$ =	41.90 " "
earths	$\frac{8.5}{68.85}$ =	$\frac{12.35}{100.00}$ " "

A slag of this nature, containing almost 46 per cent silica, although entirely feasible, is yet too silicious to melt easily and rapidly, and it would be decidedly more economical to add enough limestone (which is, ordinarily, the cheapest and most suitable flux) to reduce its content in SiO_2 to about 40 per cent. The saving thus effected in coke, as well as in ease and rapidity of smelting, would far more than offset the cost of the limestone.

Limestone consists of $CaCO_3$ plus various foreign substances — both bases and acids. As we are now looking for bases with which to reduce the silica content of our ore-mixture, we cannot tell how much limestone we need to use for the purpose until we know what proportion of bases our particular limestone contains.

Careful determinations establish that the analysis of the average limestone at our disposition is:

SiO_2	3.2 per cent
FeO	3.9 " "
CaO	39.4 " "
MgO	9.7 " "
	56.2 " "

The remaining 43.8 per cent consists of a little oxygen belonging to the iron (which is supposed to be present as Fe_2O_3), and of the carbon dioxide with which the lime and magnesia are combined. As none of these substances enter either the slag or the matte, they do not interest us.

As we are dispensing with refinements in this simple calculation, we may shorten this analysis by grouping the slag-forming constituents of the limestone under two heads:

SiO_2	3.2	per cent
bases	53.0	" "
	56.2	" "

We see at once, however, that the limestone is not an unalloyed benefit to our mixture, for it contains the unwelcome silica as well as the useful bases; and whenever we charge 100 pounds of limestone into the furnace for the purpose of obtaining its 53 lb. of bases, we are obliged to take 3.2 lb. of the injurious silica along with it. This could, of course, be allowed for in figuring the results, but would introduce a decided complication into our calculations every time that a little limestone was added, and smelters have adopted the expedient of calculating, once for all time, how far this unwelcome silica will go toward neutralizing the valuable bases, and then calling the net bases which remain — the "available bases" of the limestone. As this point comes up constantly, and with many kinds of material, in metallurgical calculations, it is worth while to explain it in detail.

As we intend to make a mixture that shall yield a slag containing just 40 per cent of silica — and, consequently, 60 per cent bases — it is clear that we must supply each pound of silica with $\frac{60}{40} = 1.5$ lb. of bases. If we then supply the 3.2 lb. of silica (contained in 100 lb. limestone) with $3.2 \times 1.5 = 4.8$ lb. bases, we shall have cleared it away entirely, so far as our calculations are concerned. We have got rid of the 3.2 lb. silica by canceling it against 4.8 of the limestone bases, and have, of course, reduced our supply of bases by just this amount. Making these deductions, we have, finally, $53 - 4.8 = 48.2$ lb. of useful bases in 100 lb. of limestone; or, as a more convenient statement for purposes of calculation, we may say that to obtain 1 lb. of available bases, we must use $\frac{100}{48.2} = 2.075$ lb. of limestone.

This 2.075 is a permanent factor in this case, and the calcu-

lation need never be repeated so long as the composition of the limestone remains the same.

Returning to the table of slag-forming constituents in 100 lb. of roasted ore (p. 184), to which we desire to add sufficient limestone to produce a slag containing exactly 40 per cent silica, we see at once that the problem is one of extreme simplicity.

We have 31.5 lb. silica, which is to be 40 per cent of the weight of our new slag; therefore, the total weight of the new slag must be $\frac{31.5}{40} \times 100 = 78.75$ lb. Of this amount, 31.5 lb. is silica, and the rest must be bases. Consequently, we must have 78.75 − 31.5 = 47.25 lb. of bases. As shown by the table, we have already 37.35 lb. of bases (28.85 lb. FeO + 8.5 lb. earths), so that we must add 47.25 − 37.35 = 9.9 lb. of bases.

Lime being the base selected, we know that in order to obtain 1 lb. of available lime we have to use 2.075 lb. limestone, or a total of 9.9 × 2.075 = 20.54 lb. of limestone.

This being added, the slag-forming constituents of 100 pounds of the mixture will consist of

silica	31.5
bases	47.25
	78.75

Expressed in percentage, this slag consists of

silica	$\frac{31.5}{78.75}$	= 40	per cent
bases	$\frac{47.25}{78.75}$	= $\frac{60}{100}$	" "

and thus fulfils our requirements.

This simple illustration must suffice to indicate the manner in which the metallurgist determines in advance the approximate composition of the slag and matte which should result from the smelting of a given mixture. A thorough comprehension of the principles upon which it is based is all that is necessary to enable him to calculate the most complicated problems of this class that are ever likely to occur in practical work.[1]

[1] See "Principles of Copper Smelting," pp. 118–141, for a complicated illustration of this kind in which all of the steps are explained minutely.

An apparent complication is introduced when the furnace-mixture is made up of several different ores and fluxes, as is usually the case in actual work; but this difficulty is removed at the outset by merely determining the weight of each constituent of the entire mixture, and thus treating it as though it were a single ore. This is too simple a matter to demand specific instructions, but experience has evolved certain methods of accomplishing this result in the shortest and easiest manner, and I therefore append an illustration of the general method pursued in smelting plants.

Three ores — *a*, *b*, and *c* — are received from three different mines in the following proportion:

a ore	200 tons per day
b ore	300 tons per day
c ore	500 tons per day
Total	1000 tons per day

If smelted together in this same proportion, what would be the weight and assay of the matte, and what would be the composition of the slag? Assume the same metallurgical losses as in the preceding case: namely, copper, five per cent; silver, six per cent; gold, two per cent; sulphur, twelve per cent.

The following analyses give the slag-forming and matte-forming constitutents of the three ores. The assayer at the smelter has no time for determining constituents which have no bearing on the result.

ANALYSES

	A	B	C
SiO_2	62.0	31.7	24.2
FeO + MnO	16.6	14.3	48.3
CaO	3.0	21.4	—
MgO	1.4	3.1	—
Al_2O_3	8.8	4.4	2.7
Cu	—	8.4	9.4
S	—	11.1	6.6
Ag	— 38.2 oz. p. ton	— 2 oz. p. ton	— 1.8 oz. p. ton
Au	— 0.1 oz. p. ton	—	— 0.3 oz. p. ton
	91.8	94.4	91.2

According to the problem, we are to smelt daily a mixture consisting of 200 tons of *a*, 300 tons of *b*, and 500 tons of *c*. For convenience, we change these large numbers to pounds, and base the calculation upon 100 lb. of the mixture, which, therefore, will consist of 20 lb. of *a*, 30 lb. of *b*, and 50 lb. of *c*.

The following table indicates merely a convenient and customary form of making this calculation, the sole object of which is to find what weight of each substance is contained in the mixture.[1]

Name of Ore	Pounds of Ore	SiO_2		FeO		Al_2O_3		CaO		MgO		Cu		S		After deducting metallurgical losses	
		Per cent	Pounds	Per cent	Pounds	Per cent	Pounds	Per cent	Pounds	Per cent	Pounds	Per cent	Pounds	Per cent	Pounds	Cu. lb.	S. lb.
a	20	62.0	12.4	16.6	3.32	8.8	1.76	3.0	0.6	1.4	0.28						
b	30	31.7	9.51	14.3	4.29	4.4	1.32	21.4	6.42	3.1	0.93	8.4	2.52	11.1	3.33		
c	50	24.2	12.1	48.3	24.15	2.7	1.35					9.4	4.7	6.6	3.3		
Total	100		34.01		31.76		4.43		7.02		1.21		7.22		6.63	6.86	5.83
			Deduct for matte		9.26												
			Net FeO		22.5												

We have now determined the slag- and matte-forming constituents of our mixture of the three ores in their required proportions, and the illustration becomes practically the same as the preceding one. I will, however, complete it here concisely, referring the reader to the former one for any details which may not be obvious.

The first step is to determine the composition of the matte, in order to know how much FeO there will be remaining for the slag.

The Cu weighs 6.86 lb., and will require $6.86 \times 0.25 = 1.715$ lb. S to form Cu_2S, leaving $5.83 - 1.715 = 4.115$ lb. S. This will take up $4.115 \times 1.75 = 7.2$ lb. Fe to form FeS.

[1] It is simpler to calculate separately the values of the precious metals. As they are assumed to have no weight, it is not necessary to include them among the constituents of the slag or matte.

Adding together the constituents of the matte, we have

copper .....6.86 = $\frac{6.86}{19.9}$ = 34.47 per cent

sulphur for copper.....1.715
sulphur remaining.....4.115
iron for sulphur 7.2

Weight of matte from 100 lb. ore ....19.9 lb.

Ratio of concentration $\frac{100}{19.9}$ = 5.025 to 1

Knowing the weight of the iron which has gone into the matte (7.2 lb.), we are now in position to determine the composition of the slag. The 7.2 lb. Fe expressed as FeO will be $7.2 \times \frac{9}{7} = 9.26$ lb. This we deduct from the total FeO of the mixture and have, finally, the exact amount of all the slag-forming constituents contained in the hundred pounds of mixed ores.

SLAG-FORMING CONSTITUENTS IN 100 LB. Mixed ORES

	lb.		per cent[1]
SiO_2 .....34.01	= $\frac{34.01}{69.17}$	=	49.17
FeO.....22.5	= $\frac{22.5}{69.17}$	=	32.53
Al_2O_3..... 4.43	= $\frac{4.43}{69.17}$	=	6.4
CaO..... 7.02	= $\frac{7.02}{69.17}$	=	10.15
MgO ..... $\frac{1.21}{69.17}$	= $\frac{1.21}{69.17}$	=	$\frac{1.75}{100.00}$

[1] This slag is much too high in silica for any ordinary work, while the matte is rather low in copper for rapid and economical converting.

The trouble is, obviously, that too much iron has gone into the matte, and has thus been withdrawn from the slag where it would have been so useful; this means that there is too much sulphur in the ore-mixture.

In actual practice, and assuming that the supply of the three different ores remains relatively the same, there are three alternative courses which might be adopted to remedy the situation, and the reason that a metallurgist is paid a high salary is because he is a man capable of deciding which of these courses is the most advantageous in any given situation. These three plans are:

1. The addition of sufficient limestone to the mixture to bring the percentage of silica in the slag down to a comfortable figure — perhaps 42 per cent. It is most unreasonable to add expensive basic flux when the mixture itself contains a lot of potential basic flux (iron) which is being worse than

The value of the matte in gold and silver is yet needed to complete the problem.

In making this portion of the calculation which deals with such minute quantities, and which is required to express their amount in ounces per ton, it is decidedly simpler to go back to the larger figures which represent the daily delivery of the ore in tons; otherwise, if we adopt the 100-pound mixture as a basis, the silver and gold content will be represented by decimals of an ounce which must be carried out to four or five places to ensure the necessary accuracy.

ASSAY OF THE MIXTURE PER TON IN SILVER

200 tons *a* ore at 38.2 oz. per ton	= 7640 oz.
300 tons *b* ore at 2.0 oz. per ton	= 600 "
500 tons *c* ore at 1.8 oz. per ton	= 900 "
total silver content of the thousand tons	= 9140 oz.

$$\text{assay per ton of the mixture } \frac{9140}{1000} = 9.14 \text{ oz. silver}$$

The loss of silver during smelting being six per cent, the corrected assay will be $9.14 \times 0.94 = 8.59$ oz. silver per ton of smelting-mixture.

wasted. The addition of a certain amount of limestone may be inevitable, but not until the iron has been utilized to its fullest extent.

2. Roasting the whole, or a portion, of (*b*) ore which is evidently the cause of the trouble, as may be seen from the analysis. This would be an exceedingly disadvantageous plan, as the ore only carries 11.1 per cent of sulphur anyway, and would ill repay the cost of roasting, apart from the investment in plant. Indeed, how would one proceed to roast it? It does not contain sulphur enough for heap roasting; and, as may be inferred from its composition, it is pretty certainly in lump form. This would mean crushing it and unfitting it for blast-furnace smelting. The proposition is entirely out of the question.

3. The adoption of partial pyrite smelting, by which the ore would not be roasted, the coke would be reduced to the smallest possible quantity, and sufficient blast would be employed to bessemerize the melted sulphides to a point sufficient to produce a matte of the desired grade.

Under skilled management, four distinct advantages would result from this plan:

a. Saving of coke.

b. Elimination of roasting.

c. Production of a suitable matte — perhaps 40 per cent copper.

d. Utilization — as a slag-base — of the superfluous iron which entered the matte in the process given in the illustration.

corrected assay of ore × ratio of concentration = assay of matte
8.59 × 5.025 = 43.16 oz. silver per ton

ASSAY OF THE MIXTURE PER TON IN GOLD

200 tons *a* ore at 0.1 oz. per ton ..... = 20.0 oz.
300 tons *d* ore at no oz. per ton ..... = —— "
500 tons *c* ore at 0.3 oz. per ton..... = 150.0 "
total gold content of the thousand tons..... = 170.0 "

assay per ton of the mixture $\frac{170}{1000}$ = 0.17 oz. gold

The loss of gold during smelting being two per cent, the corrected assay will be 0.17 × 5.025 × 0.98 = 0.837 oz. gold per ton.

Thus the matte should assay per ton 43.16 oz. silver and 0.837 oz. gold.

Returning to the actual smelting of an oxidized charge in the coke blast-furnace, we note that the subject divides itself naturally into two parts:

1. Smelting natural oxide ores for the production of metallic copper.
2. Smelting roasted sulphide ores for the production of matte.[1]

[1] To these two important divisions might be added a third, which is so rare and so simple that it demands only this casual mention. I refer to the fusion of raw pyrite ores with coke (or charcoal), with no attempt at oxidation of the sulphides and, consequently, with the production of a large proportion of low-grade matte. This is Percy's "pyritic smelting," and its entire scope and principle is the very antipodes of our modern "pyrite smelting."

As the process is conducted with a light blast and abundant coke and, furthermore, as there is no fixed oxygen (in the shape of iron or copper oxides) to react with the sulphides, the process is strictly a reducing one, and the effect of the operation — so far as concentration is concerned — is limited to the removal of the gangue-rock (usually demanding much basic flux), and the sublimation of such fraction of the sulphur as is driven off by heat alone; consequently the matte will, under ordinary conditions, consist mainly of FeS, will be low in copper, gold, and silver, and large in quantity.

Under present conditions this "Percy's pyritic smelting" would seldom be an advantageous operation. Its characteristic mission would seem to be to conserve the FeS content of a precious-metal ore too low in pyrite for true pyrite smelting, and thus turn it into a valuable, concentrated flux and fuel (FeS) for use with certain other more important pyrite ores which did not of themselves contain quite enough sulphides for pyrite smelting.

In 1905–1906 this rare process was used, in a slightly modified form, at Butte in connection with the Baggaley experiments, to furnish the low-grade matte required for the subsequent smelting of silicious ores in the

1. *Smelting natural oxide ores for the production of metallic copper.* — This process is now somewhat rare, for the simple reason that the rich oxidized ores of copper have become pretty well exhausted, and such bodies of them as are still left are drawn upon sparingly to mix with the deeper sulphide ores, and thus gain the benefit of their fixed oxygen to which I have several times referred.

As cases will, however, occur in new districts where it will seem advisable to smelt oxide ores in the blast-furnace direct for metallic copper, I will mention briefly the more important guiding principles in this operation.

The first point worthy of attention is the appreciation of the fact that in producing metallic copper the whole lower portion of the furnace and its accessories (spout, forehearth, etc.) tend to chill rather than to cut. Metallic copper is an extraordinary conductor of heat and has none of the fiery and corrosive qualities which distinguish a matte containing a considerable proportion of ferrous sulphide.

Consequently, instead of attempting to cool the hearth of the furnace by extending the water-jackets well below the tuyeres, making a thin bottom, and running the molten products as soon as formed into an exterior settler, the exact reverse is indicated.

The water-jacket should extend only just far enough below the tuyeres to offer facilities for removing scale and sediment from the water-space; the shallow hearth should be replaced by a deep, internal crucible surrounded by thick brickwork; and the outside forehearth should be reserved for the slag alone, the copper being retained in the inside crucible, and tapped periodically.

A second point in both the construction and the management of the furnace relates to its degree of reducing action. This is a delicate matter because, if the reducing effect is too strong, there will be a considerable formation of metallic iron from the iron oxides of the ore, and a corresponding formation of a most disagreeable iron-copper alloy. In the earlier furnaces of the south-

Baggaley basic-lined converter. As a rule, however, such ores as would be suitable for "Percy's pyritic smelting" can now be more advantageously treated by the partial pyrite process with production of high lime-silica slags. (For a fuller description of Baggaley's interesting work see "Principles of Copper Smelting," pp. 464–467.)

west I have seen black copper (which is the ordinary name for the metallic copper produced from smelting oxide ores in the blast-furnace) which contained only about 60 per cent of copper, most of the remaining 40 per cent consisting of metallic iron. Such a product not only entails extravagant refining charges, but also robs the slag of its fluxing-iron and, furthermore, separates badly, causing a slag rich in copper.

This seems to be a sufficient train of evils to result from a single fault, but, in reality, there is worse to come. The slag, being robbed of several per cent of its FeO, becomes too silicious and infusible, runs sticky and slow, and invariably induces the foreman to add more coke "to heat up the furnace." The extra coke enhances greatly the reducing power of the furnace, and this, with the increased temperature and the slow sinking of the charge, brings about the conditions of a furnace making pig-iron from ore, and soon necessitates blowing out.

Insufficient reduction causes slagging of copper oxides, but is less common and less difficult to deal with than the opposite condition. That a happy medium is not too difficult to demand is evinced by the regular results attained for years at the Bisbee and Globe (Arizona) plants when they were smelting oxidized ores.[1]

The next most important point to consider in the smelting of oxide ores is the loss of copper in the slag. This is always high — as compared with matte smelting — but is frequently higher than it need be.[2]

[1] Dr. Trippel gives an analysis of the average sample of two weeks' production of blast-furnace copper from his own smelting, at Globe (Arizona), of oxide ores having the following composition:

Ore		Black Copper	
Silica	20.23	Copper	99.11
Ferric oxide	42.10	Lead	0.67
Alumina	4.15	Sulphur	0.08
Loss by ignition	9.75	Slag	0.08
Oxide of copper	17.12	Arsenic	Trace
Magnesia	2.85	Iron	Trace
Lime	1.12		99.94
Oxide of manganese.	1.63		
	98.95		

[2] Cooper demonstrated that the use of anthracite coal instead of coke produced much cleaner slags in this kind of smelting.

The slag losses arise from one of two causes:

(*a*) copper slagged as oxide, and usually combined with silica.

(*b*) copper carried away mechanically in metallic globules.

(*a*) Copper slagged as oxide. — This is usually small as compared with the losses under division (*b*). Still, as already intimated, a feeble reducing action in the furnace may allow some of the copper to remain as oxide until it sinks to a zone where the temperature is high enough to awaken affinity between itself and silica.

Also when the coke is charged in too large fragments, or the individual charges of coke and ore are too great in volume, there may exist in the furnace a circumscribed, heated area near the tuyeres, which has an oxidizing or neutral, rather than a reducing, atmosphere, and this may cause a considerable formation of copper silicate. The remedy is obvious: namely, increase of the reducing action. This may be effected by heightening the ore-column; by diminishing the weight of each individual charge of ore and fuel, so that the horizontal layers of these substances in the furnace-shaft are thinner, the ore thus coming more immediately in contact with the coke; by lessening the size of the lumps of coke; by making the mixture slightly more silicious, which will usually raise its melting-point, and thus elevate the temperature and increase the reducing action.

(*b*) Copper carried away mechanically in globules.— Even with a hot and well-designed interior crucible, the slag will contain more or less metallic copper in the shape of minute prills or globules. This will always demand careful, and often elaborate, settling devices to separate it satisfactorily.[1]

2. *Smelting roasted sulphide ores for the production of matte.* — This division brings us back again to the old-fashioned use of the blast-furnace as a melter-down of partly-oxidized (roasted) sulphide ores of copper and iron.

As already intimated, this process has become greatly circumscribed since the opening of this century, because we have learned that it is possible to utilize so considerable a proportion of the heat evolved by the rapid oxidation of the sulphides that, instead of being a source of expense as having to undergo a costly

[1] For fuller details regarding this subject, see "Principles of Copper Smelting," pp. 141–145.

roasting, they now become a valuable fuel and, under certain conditions, may replace coke to an extent out of all proportion to their actual calorific value.[1]

It follows that nearly all of our blast-furnace plants are taking advantage of this new practice, and are smelting their sulphide ores without roasting. This does not mean that the country is filled with "true pyrite-smelting" plants; quite the contrary. True pyrite smelting demands those great lenses of massive cupriferous pyrite which are among the rarest of all ore-bodies.

The process that at present characterizes the copper blast-furnace practice of North America is "partial pyrite smelting," in which the sulphides are used to supplement the coke (or *vice versa*), and *blast enough is employed to burn both.* Consequently, partial pyrite smelting represents the typical blast-furnace practice of the American metallurgist, and must be treated correspondingly.

The brief space which I can afford for the consideration of the old-fashioned method must be devoted to a study of its aims and its reactions, rather than of its actual results. We do not care to study the conditions of furnaces, management, trans-

[1] A pound of coke has, of course, a much higher calorific value than a pound of sulphur (with its accompanying Fe, etc.); yet, under certain conditions, as much as one-third of an already moderate coke charge may be replaced by a similar weight of sulphur (in the shape of pyritic concentrates) without diminishing the capacity or disturbing the running of the furnace.

While I have data from various independent sources confirming this proposition in a general way, I can offer one authoritative instance based upon scientific observations on a very large scale. Mr. Frederick Laist, Assistant Superintendent of the Washoe Smelter, informs me that, while making a series of careful experiments upon the Anaconda blast-furnaces with a view of substituting sulphides for coke so far as practicable, it was found that so long as the combined weight of the coke and sulphur in the charge reached 18 per cent of the mixture of ores and flux, it made no perceptible difference in the behavior of the furnace whether the coke was as low as 7 per cent or as high as 11 per cent. In other words, under the conditions prevailing in the Anaconda furnaces (which included ample blast), the effect of 7 coke + 11 sulphur (as pyrite, etc.) = 11 coke + 7 sulphur.

This remarkable result is due, probably, to two principal causes: first, the increased fusibility (flowing-quality, not formation-temperature) of the slag, owing to its greater FeO content, derived from the augmented pyrite; second, the more intense localization, and consequent more complete utilization, of the heat derived from the bessemerizing of the sulphides.

portation, etc., as they existed 10 or 15 years ago; yet we cannot study a discarded process with a modern environment.

This older practice is characterized by the simplicity of its reactions, and the ample margin of heat that could always be provided in cases of emergency.

The first clause of this proposition scarcely demands defense. Assuming that we started originally with raw sulphides, and finished up with matte and slag, it is manifest that we used to relieve the blast-furnace of most of the delicate and critical work, and demand of it simply that it should act as a melter-down of inert material, already prepared for it.[1]

The delicate part of the operation was the preliminary roasting and, as a matter of fact, this was not delicate either, as there is no especial difficulty in roasting material when it is in solid form, at a moderate temperature, in a highly oxidizing atmosphere, and with an absence of all harassing side-issues. With suitable ores, a good roasting ensured an easy and successful smelting. In 1884 I wrote in my first book on copper smelting — "the key to the smelting process is carried in the pocket of the roaster foreman," meaning, of course, that if the sulphides are thoroughly oxidized, the matte will be of good grade, there will be ample FeO to flux the silica, the slag will be fusible and rapid, and the entire process will be comfortable and successful.

The second clause of the proposition — stating that, in the old process, an ample margin of heat could always be provided quickly — refers, of course, to the fact that, in those cases where the tuyeres begin to grow dark and the slag cold, prompt relief could be given by simply adding a little more coke, without demoralizing the entire smelting operation as is the case in our modern practice.

In this latter process there are so many sensitive reactions involved, and there is such a delicate balance between oxidation and reduction, that any so rude disturbance as the addition of coke would disorganize the entire operation. This more com-

[1] The single exception to this statement is the fact that we expected the blast-furnace to reduce the higher oxides of iron to FeO in order that this latter substance might combine with the silica. This duty, however, demanded neither care nor attention under any ordinary circumstances, and, with a reasonably fusible slag to ensure rapid running and the immediate removal of the molten products from the interior of the furnace, there was no tendency toward the production of metallic iron.

plicated, and infinitely more difficult, process is correctly termed "reaction smelting" to show that its success depends upon suitable important reactions (mainly, oxidation of the sulphides *at the right moment*) which are carried out in the blast-furnace, while the older process has had all these reactions executed for it in the preliminary operation of roasting.

As I have already pointed out, in this straight coke-smelting process the column of charge within the furnace-shaft rests, normally, upon a permanent bed of incandescent coke which exists in the space below the tuyeres. This bed may be regarded as a highly-heated grating through which the charge is strained, as through a filter, only the liquid portion being able to pass it. The easily-fused constituents of the mixture liquate out at a considerable distance above the tuyeres, and drain down rapidly through the coke grating. The more refractory compounds, on the contrary, are held up at this point until one of three things occurs: either, (*a*) they take up other substances and form new compounds which melt at the established normal temperature of the fusion-zone (say, 1375 deg. C.) and pass the coke barrier safely; or, (*b*) they remain stationary, blocking the descent of the charge, while the combustion of the coke at, and above, the tuyeres still continues. As increased heat is being produced, without being absorbed by the normal cooling descent of the ore-column, the temperature rises. When it reaches, perhaps, 1450 deg. C., the blockade is raised. The melting-point of the refractory compounds resting upon the coke grating is reached; they melt, drain away as before, and the ore-column resumes its regular descent.

Everything about the furnace may appear as before; yet there is one important difference: the normal temperature of the smelting-zone has been raised by 75 deg. C., and is now 1450 instead of 1375 deg. C. So long as this new normal temperature is maintained the process will continue without interruption, although requiring a larger proportion of coke than with the lower heat; or, (*c*) there is not enough of the indispensable foreign substance (see (*a*)) to form 1450-degree compounds with the refractory constituents, so that only a part of the latter can be melted and drained away, and the undigested portion will remain in the region of the tuyeres in a semi-fused condition and awaiting either more flux, or the still higher temperature which it demands.

If neither of these is provided, the undigested material accumu-

lates and the ore-column again ceases its descent, although the coke which is mixed through it still continues to burn. The furnace is freezing up and, as soon as the coke of the fusion-zone has burned away, the wind blows the furnace cold instead of hot.

As a matter of course, in actual practice there are too many side-influences at work to permit the evolution of all these phenomena in so simple and systematic a form as they appear on paper. I believe, however, that this method of picturing in one's imagination the phenomena of the coke-burning blast-furnace process is the most useful means at our disposition of obtaining a correct idea of its operation and of its needs. Taken in connection with the cancellation of such constituents as neutralize one another, it leads to a clear and logical grasp of the general situation.

For instance, imagining a mixture of the same nature as the one in the preceding illustration, and assuming that a slag with 40 per cent silica and 60 per cent bases will suit our needs and will correspond to the degree of heat which we desire to establish as the normal temperature of our smelting-zone, we may simplify the situation as follows. Beginning with the matte, we cancel the copper of the ore-mixture against the sulphur in the proportion of 1 to 0.25.[1]

Next we cancel each pound of the remaining sulphur against 1.75 lb. iron. The matte is now disposed of, and may be conceived as having liquated out of the ore, passed through the coke grating and, consequently, disappeared from our consideration.

Knowing now how much Fe remains to act as a base (FeO), we may proceed to the cancellation of the silica against the bases. Forty pounds of silica is to combine with sixty pounds of bases. Consequently, 1 lb. silica cancels 1.5 lb. bases.

If the silica and bases cancel in this proportion without any remainder, we know that our ore-mixture is correct, and that it will melt completely at the normal temperature of the fusion-zone, and pass through the coke-grating without remainder.

If, however, there is a remainder of either silica or bases after the cancellation, it must be looked to immediately. Either there

[1] See preceding illustration for detailed explanations of these figures and operations.

must be added to the mixture enough of the required substance to cancel this remainder, or else we must change the condition of the smelting-zone (the temperature) to such an extent that a slag may form which shall take up this uncanceled remnant and which, therefore, will not consist of 40 per cent silica and 60 per cent bases, but might — for instance — contain 35 per cent silica and 65 per cent bases, or 45 per cent silica and 55 per cent bases, etc.

The knowledge and experience which enable the person responsible for this work to decide whether it will be more advantageous to maintain his normal furnace-conditions and change the composition of his slag to conform to them, by adding the lacking substance, or whether it will be better to rearrange his furnace-conditions so that they shall correspond to such a slag as his existing ore-mixture may happen to yield — this, and the ability to execute these decisions, represent the class of problems with which the metallurgist is called upon constantly to deal.[1]

I am particularly anxious to make this subject clear in order that the reader may appreciate the extraordinary difference that obtains between the ordinary coke-smelting process and the non-coke-smelting operation which we term "true pyrite smelting." It is not until the student has a complete and correct conception of these two widely divergent type-forms that he is in a position to understand and to practise the peculiar hybrid process which is the most difficult, far-reaching, and commercially important of all copper smelting operations: namely, "partial pyrite smelting."

Anticipating the subject for a moment, let me, in recapitulation, point out concisely the remarkable contrast between the two methods.[2]

[1] While it is convenient at the outset to simplify the study of slags by regarding them merely as compounds of two opposing substances — silica and bases — I must again point out that it is by no means a matter of indifference what these various oxides may be that we have, for convenience, lumped together under the generic name of "bases." This matter will be taken up more particularly in the chapter on "Slags."

[2] In any proposition based upon *contrast*, it is, of course, essential to dwell more particularly upon those points which present the most striking differences.

Coke smelting of roasted ores

Constituents of ore-mixture inert and incapable of evolving heat.

Constituents of mixture oxidized and already in proper chemical condition to combine, providing they are furnished with the necessary heat from some outside source (coke).[1]

The percentage of silica in the slag is (within wide limits) at the pleasure of the operator, as he can increase his heat at will, and thus cause unwilling substances to combine (or dissolve), and thus produce a practicable slag from comparatively unsuitable substances.

The degree of concentration depends upon the completeness of the preliminary roasting, and is already established before the mixture is fed into the furnace.

True pyrite smelting [1]

Constituents of charge active and capable of producing all the heat necessary for fusion.

Basic constituents of charge unoxidized, and combined with sulphur. Can combine with silica only after being liberated and modified in the smelting process itself.

The percentage of silica in the slag is determined by the process itself, as it is fixed by the degree of intensity of the bessemerizing reactions, and there is no exterior source of heat which may be drawn upon to force indifferent or unwilling ingredients to unite into a flowable liquid.

The degree of concentration depends solely upon the completeness of the bessemerizing reactions in the furnace itself.

RECAPITULATION

A mere fusion of comparatively inert constituents which have undergone preparation in advance and require only the application of heat to melt into

An opportunity offered to certain incongruous substances to carry out a series of reactions, by means of which they shall generate sufficient heat to main-

[1] To be sure, the iron oxides in the roasted ore have to be reduced to FeO to fit them for our purposes; but this is a destructive rather than a constructive reaction and, moreover, is practically automatic.

[1] Assumed to be conducted absolutely without coke, this incorrect assumption being legitimate in the present argument, as the small amount of coke used never reaches the zone of fusion, and thus has no influence upon the features of the process which interest us for the moment.

two portions: a new sulphide of metals, corresponding to the amount of sulphur left in the ore; and an indefinite fused mixture of several silica-salts, often holding excess metal-oxides in solution.

I do not think that any one who has given especial study to the intimate composition of ordinary copper slags will take exception to my characterization of many of them as "indefinite compounds." They may serve their purpose admirably; but it certainly requires more than justifiable ingenuity to provide them with a chemical formula.

tain the desired temperature, shall lose their incongruity and develop affinity, and shall eventually form a specific silicate-salt of a composition corresponding to the temperature which they were able to evolve, and which shall be liquid enough to flow out of the furnace.

Incidentally, the excess sulphides — which failed to take part in the reaction-stage — escape as matte.

The substitution of wood for a certain proportion of the coke in blast-furnace smelting is a matter that is not likely to be of interest to the managers of large well-provided smelters in accessible situations; but the metallurgist who is trying to keep a small plant running in a remote district will find it well worth his trouble to give more attention to this subject than he is ordinarily in the habit of bestowing upon it.

The most favorable account of the employment of wood in this manner with which I am familiar is the complete substitution of wood for coke at the smelter of the Mitchell Mining Company, Guerrero, Mexico, reported by its president.[1]

The 200-ton plant was doing partial pyrite smelting on raw copper-sulphide ores, using coke as a fuel and making a 40 per cent matte. Being in a forest district, and remote from the railway, experiments were tried in substituting oak wood and charcoal for a portion of the coke. One-third each of oak blocks, charcoal, and coke was not satisfactory, but two-thirds wood and one-third coke melted the charge well enough, but yielded a matte with only 28 to 30 per cent copper from the same charge that produced a 40 per cent matte when smelted with straight

[1] "Use of Wood in Copper Smelting," by George Mitchell, *Engineering and Mining Journal*, Oct. 13, 1906. I make use of this report in describing the situation.

coke. It was found that the three-inch wooden blocks ignited near the top and burned out before reaching the fusion zone.

This difficulty was greatly diminished by dipping the wooden blocks in a thin paste of silica and clay. The proportions of this paste, or wash, were such that there was just enough clay present to act as a binder so that the coating would not crack and fall off. The charge in this instance was basic, but Mr. Mitchell advises that, for a silicious charge, lime should be substituted for the silica so far as possible.

It was found that blocks treated in this manner passed nearly down to the fusion-zone before catching fire, thus obviating the hot top, and carrying their carbon to the point where it was required. The results were so good that the proportion of coke was gradually diminished until it was eliminated completely. The copper in the matte is kept at about 38 per cent as against 20 to 25 per cent when uncoated wood is used, and it was the intention of the company to smelt exclusively in this manner until the completion of their railway. The capacity of the furnace is considerably less than with coke, but the costs per ton are low, owing to the cheapness of the fuel.[1]

In 1901, Collins substituted wood for a certain proportion of his coke at[2] Santa Fe, Chiapas, Mexico. The ore consisted of bricked copper concentrates, coarse garnet jig middlings, and selected garnet ore, carrying chalcopyrite and bornite. About 15 per cent of coke was consumed in smelting the charge, and this was reduced to 11½ per cent by the use of wood in pieces of considerable length. It was found that not more than 25 per cent of the coke could be replaced by wood without causing a hot top with its attendant troubles. Contrary to Mitchell's experience in the preceding case, his matte increased nearly five per cent in copper, averaging about fifty per cent.

Bretherton reports fair success in the substitution of wood

[1] This result is so much more favorable than I have ever seen recorded, or have ever experienced myself when trying to eke out a scanty coke supply by the use of wood, it seems that circumstances at the above plant must have been peculiarly suited to such practice. I must admit, however, that I have never protected the surface of the wood, or seen it protected, in the manner described by Mr. Mitchell, and to which he ascribes so much of his success.

[2] See paper read by H. F. Collins before the Institution of Mining and Metallurgy, October, 1902.

for a portion [1] of his coke in smelting the particularly difficult ores of the Great Western Gold Mining Company, Ingot, California.

Owing to a shortage of coke, he replaced about one-half of this substance with short blocks of green live oak and "Digger pine" — both superior fuels — using two pounds of wood for each pound of coke taken off. This wood lasted well down toward the fusion-zone, but would not stand much pressure of blast without causing a hot top and a corresponding chilling of the crucible.

That this success was due mainly to the superior quality of the wood was shown by the fact that subsequent trials with inferior wood — at times coated with a wash, as in Mitchell's case — were not at all favorable, the crucible beginning to chill within a few hours.

My own experience emphasizes the importance of green wood of as massive a character as possible, of a light blast, and, consequently, of modest expectations as to capacity of furnace.

[1] "Use of Wood in Matte-Smelting, together with Results of a new Hot-Blast Stove," by S. E. Bretherton. *Engineering and Mining Journal*, Dec. 1, 1906.

CHAPTER VIII

TRUE PYRITE SMELTING

THE most important blast-furnace method for the copper smelter of the present day is, undoubtedly, *partial pyrite smelting*, by which term is meant the fusion of moderately pyritic ores in the blast-furnace, with sufficient wind to burn not only the moderate amount of coke used in the operation, but also a certain proportion of the sulphides present, thus obviating any preliminary roasting, and saving coke by utilizing the heat evolved by the combustion of the sulphides.

This process is a compromise between ordinary coke smelting and true pyrite smelting, although much more closely allied to the latter.

The coke-smelting process has been studied in a preceding chapter, and it now remains to examine the principles which underlie true pyrite smelting, as a preparation for the comprehension of the commercially more important hybrid, partial pyrite smelting.[1]

I think that the best way to obtain a true conception of the phenomena of pyrite smelting is to divest the mind completely of all idea of roasting or gradual preparation of the ore which is sinking in the shaft of the pyrite furnace. As the mind craves some familiar garb in which to clothe the reactions that take place within the furnace-shaft, I believe that the most suitable image which one can picture to himself is that of the column of sulphides and quartz sinking unchanged in the furnace, and still remaining as sulphides and quartz until they reach such a tem-

[1] The principles and practice of pyrite smelting have been studied at unusual length in my "Principles of Copper Smelting," where I have had the invaluable aid of Robert Sticht, General Manager of the Mount Lyell Mining & Railway Co., of Tasmania.

The purposes of the present book are of a different nature, and I must compress the subject into a space entirely inadequate to its scope and importance.

perature that the sulphides liquate away from the infusible gangue, gain the oxidizing-zone where they are bessemerized instantaneously and, remaining now only as oxidized iron which cannot exist as FeO without silica, proceed to eat away the foot of the quartz column as the ferrous oxide eats away the lining of the converter. (Incidentally, such melted sulphides as failed of bessemerizing in the oxidizing-zone drop to the bottom of the hearth as *matte*.)

While this conception will require considerable modification and elucidation before it can be accepted as a strictly scientific presentation of facts, it still comes very near the truth and, if fostered, will at least substitute a correct outline for the totally false idea of a gradual roasting and preparation of the sulphide ores in the shaft of the pyrite furnace which is still too prevalent.

Even in true pyrite smelting, practised on characteristic, massive sulphide ores, a little coke is found advantageous; but — and this simplifies materially the study of the process — it does not prejudice the bessemerizing action of the smelting-zone, because it is all consumed higher up in the shaft.

I have explained in a preceding chapter that unnecessary confusion results from trying to treat partial pyrite smelting as a mere slight modification of the true pyrite process, and thus attempting to study them both at the same time. They are widely differing operations and, as a sharp division between them had never been established, I ventured in a former book to draw such a line at the boundary which nature has marked with unusual distinctness: namely, at the point where the amount of coke used becomes so large that it is not all consumed in the upper regions of the shaft, but descends into the bessemerizing-zone and begins to appropriate the oxygen which is needed for the bessemerizing of the sulphides. A clear understanding of the arguments to follow cannot be attained without constantly bearing in mind this distinction between true and partial pyrite smelting.

The most convenient way in which to begin the study of the phenomena of true pyrite smelting is to strip the process of all complications, and reduce the constituents of the furnace-mixture to the two which are absolutely essential to the existence of the process in its common form: namely, silica and an iron sulphide.

The silica may best be represented by ordinary quartz, and the iron sulphide by pyrite — the most frequent mineral of this class.

In order that we may have a familiar variety of matte to deal with, I will add one more constituent to the charge, and will assume that the pyrite contains a small amount of disseminated chalcopyrite. Excepting for the absence of a few per cent of earths, this mixture represents the main features of most of the pyrite charges which are treated.[1]

Beginning with the three familiar substances, quartz, pyrite, and chalcopyrite, let us follow them from the charging-door of the furnace until they have undergone all those changes of form and all those rearrangements of atoms and molecules which it is the duty of the pyrite furnace to effect.

As the temperature of the upper region of the shaft is comparatively low, and as silica undergoes no change either of form or of constitution until a considerable heat is attained, we may eliminate this substance from our initial observations, and confine our attention to the two sulphides.

The behavior of these minerals varies profoundly according to the condition of the atmosphere to which they are exposed during their slow descent through the progressive temperature of the shaft.

Sticht's repeated analyses of the gases of the Mt. Lyell furnaces, withdrawn from different horizons during the normal smelting operation, have given us a certain amount of positive knowledge on this important subject, and these analyses are corroborated by the condition of the charge at the corresponding horizon, as shown by samples taken therefrom.[2]

The following table exhibits Sticht's 40 analyses of the gases drawn from different horizons of the Mt. Lyell pyrite furnaces, using about 14 per cent coke.

[1] For reasons already indicated, we need not at present pay any attention to the small proportion of coke which is commonly added to the charge even in the true pyrite-smelting furnace. The fact that it is all consumed before reaching the zone of oxidation removes it from the list of constituents which interest us during this preliminary discussion.

[2] See "Principles of Copper Smelting," p. 228, for a review and discussion of these analyses. The Tilt Cove furnaces, running absolutely without coke, contained less than one per cent oxygen in the upper part of the shaft.

PERCENTAGE BY VOLUME

Number of Samples	Depth from Surface — ft.	SO_2	CO_2	CO	O
5	2 to 2½	6.64	5.08	0.16	0.84
4	2 to 2½	7.95	3.075	0.0	1.50
4	2½	8.925	5.45	0.0	0.70
13	2 to 3½	7.88	5.93	0.02	0.35
5	3 to 3½	6.12	7.86	0.0	0.66
4	6	9.475	4.7	0.0	0.0
1	6	10.60	4.40	0.2	0.0
5	6 to 7	7.90	3.56	0.0	0.83

Perhaps the most interesting feature of these analyses is the exceedingly small amount of O that they show, and the fact that the proportion of this element *does not increase as the zone of combustion is approached.*

This demonstrates how complete combustion must be at that point, and how absolutely non-oxidizing are the resulting gases, when the furnace is running normally.

This research work established beyond doubt that, during normal pyrite smelting, there is practically no oxygen in the upper regions of the shaft, and that there is no increase in oxygen until the actual zone of combustion is reached. This means that the bessemerizing (oxidizing) action is confined to a distinct and circumscribed zone (termed the focus), that all the oxygen of the blast is consumed in this zone and, consequently, that the atmosphere in the upper portion of the shaft is either neutral or reducing. As a matter of fact, it may be regarded as neutral, and the problem for our immediate examination is to establish the behavior of pyrite and chalcopyrite when subjected to progressive heat in a neutral atmosphere. This point has already been studied in Chapter IV but, for the sake of convenience, I will repeat briefly what has there been discussed in greater detail.

In an atmosphere free from oxygen, and exposed to a dull-red heat, pyrite (FeS_2) loses about $\frac{3}{7}$ of its sulphur and becomes a compound analogous to pyrrhotite (Fe_7S_8). This occurs normally within a few feet of the surface of the charge and, as the descent continues and the temperature increases, this Fe_7S_8 melts at

about 925 deg. C., and at once hurries forward by gravity to the still hotter levels below. By the time it has attained a temperature of 1200 deg. C. it has lost another fraction of its sulphur, and has now reached the lowest possible combination of iron and sulphur — FeS. It is now approaching the upper limit of the focus and, at the high temperature which prevails there, even the comparatively stable FeS undergoes a further slow dissociation (without the presence of oxygen), becoming a mixture of FeS with variable, and slowly increasing, admixture of metallic iron — the resulting substance being expressed adequately by the formula $Fe_n \cdot FeS$.

This is the true pyrite fuel of the furnace, and an appreciation of this fact is of great importance in this process, as well as in various other operations — notably in the blast-roasting of sulphide fines as a preparation for the blast-furnace.

Let us examine this statement for a moment, in order to establish the point without question. Our potential fuel at the outset consisted of FeS_2 (neglecting, for the moment, its accompanying slight admixture of chalcopyrite) and, therefore, presents two substances capable of evolving heat by oxidation. These are iron and sulphur. The iron is still unoxidized at the lowest horizon which we have attained, and can all be accounted for in the residual $Fe_n \cdot FeS$; consequently, it has done nothing toward the production of heat. The sulphur — belonging to the original FeS_2 — has been driven off by heat alone, into an atmosphere devoid of oxygen; evidently, there is no evolution of heat from this source. What really occurs is that the sublimated elemental sulphur ascends with the gases as a yellowish vapor until it reaches the surface of the charge, and can find air to burn it to SO_2. Therefore, the only action of the sulphur is to burn in the free air, on top of the charge, and produce a small amount of heat where it is not wanted. Indeed, there has been a considerable loss of heat in dissociating the FeS_2, and vaporizing the sulphur, and the entire useful effect of the pyrite is now concentrated in the $Fe_n \cdot FeS$ which is just about to enter the focus, or oxidizing-zone.

The exceedingly liquid $Fe_n \cdot FeS$, percolating the numberless interstices of the white-hot column of quartz that fills the narrowed oxidizing-zone (bessemer-slot), meets the upward rush of heated oxygen and flashes instantaneously into SO_2 gas and FeO,

the latter combining, *in statu nascendi*, with the eager acid (silica) with which it is in contact.

The conditions of the bessemer converter are present in an intensified form, as the juxtaposition of the melted sulphide and the white-hot silica is a powerful incentive to the instantaneous formation and removal of the FeO.

The oxidation of the sulphur and iron, together with the union of FeO and SiO_2, produces a local intensity of temperature that is most striking, and which is maintained by driving the process at a rate so rapid as to neutralize the losses by radiation and convection, and still provide a scanty margin of heat.[1]

It is at this slag-forming stage of the process that the distinctive features of pyrite smelting stand out in their most clear and characteristic form.

The ferrous oxide takes up just enough of the silica with which it is in contact to form a slag corresponding to the temperature which, for the moment, is reigning in the focus. That is to say, the temperature of the smelting-zone determines the amount of silica which the FeO will take up and, consequently, the silicate-degree of the slag which the furnace will produce. For a given temperature, this is practically unalterable.

You may feed more quartz in the endeavor to make a more silicious slag, but it is useless. The FeO will take up just the amount of silica necessary to make a slag of the formation temperature corresponding to the heat of the focus, and this slag, as soon as liquid, will flow away and leave the excess silica behind. Or, as is more often the case in true pyrite smelting, there is a dearth of silica; the metallurgist desires to get along with as little of this substance as possible, and thus increases his pro-

[1] This conception of pyrite smelting, based upon positive evidence, contradicts absolutely the theory of a gradual roasting of the sulphides in the upper levels of the shaft. Even with the total absence of oxidation, and the conservation of every possible atom of our fuel until it can be burned with great rapidity at the exact point where proper slag-formation is to begin, the margin of heat is small, and serves only for the normal purposes of the process. Any slight irregularity or rising of the superior boundary line of the zone of highest temperature causes a waste of resources that the process itself is seldom able to endure; and to imagine that our heat-producing substances could be frittered away by the gradual roasting of the sulphides in the upper levels of the shaft not only does violence to all experience, but credits it with heat-potentialities that, if properly utilized, would change it from one of the most difficult to one of the easiest of all smelting operations.

portion of pyrite; but if the other conditions remain unaltered, the slag will also remain the same as before, and the increased sulphide will only go to dilute the matte.

As is often said, "the pyrite furnace chooses its own slag," and this is true so far as its choosing a slag that corresponds to the temperature of the focus, and refusing to be coerced into modifying that slag in any way; differing in this respect entirely from the coke-burning furnace, where the natural affinities of the various constituents are, as one might say, stifled and overridden by the all-powerful influence of unlimited, extraneous heat.

But this dictum does not mean that the pyrite furnace is confined unalterably to one single type of slag, and will never choose any other. On the contrary, its slag-type may be varied through a range which, although circumscribed as compared with the wide latitude of coke smelting, is still broad enough to cover most of the cases which one is likely to encounter in practice; only, instead of changing the type of slag by overriding the influences within the furnace, we have to alter the furnace-influences themselves, and then the process will establish a new type-slag in harmony with the new conditions.

No more admirable illustration of this proposition is needed than the situation at Mount Lyell, when Sticht increased the height and wind-pressure of the original hot-blast pyrite furnaces and changed to cold air, for the purpose of obtaining a higher degree of concentration and a more basic, ferruginous slag. The correctness of his reasoning is shown in the results.[1]

The old Mount Lyell hot-blast furnaces represented, at the time, as good construction and management as the field of true pyrite smelting then offered, and were run with a greater volume

[1] This brief account of the first period of transition at the Mount Lyell smelter does not apply entirely to present conditions, which have been modified by changes in the ore-supply, as well as by the fact that the pure massive pyrite of the main Mount Lyell ore-body has changed its character greatly in depth, containing now considerable proportions of galena and blende.

Nevertheless, the increase of height of furnace and of volume and pressure of blast has still proceeded, the last change giving 18 ft. (5.5 m.) of ore-column between tuyeres and downcast, and indicating that even these figures may be exceeded with advantage.

I make free use of Sticht's original statements in describing this first great change from hot to cold blast, and from low to higher furnaces, as I believe it to be the most enlightening contribution extant to the practice, as well as to the theory, of true pyrite smelting.

of blast and with more rapid driving than any other of which I have knowledge.

The conditions were as follows: an unlimited supply of massive pyrite — containing a little chalcopyrite — and a scarcity of profitable quartz with which to flux it. The aim of the process was, therefore, to smelt the greatest possible quantity of iron with the least possible quantity of silica: in other words, to produce a slag of as low a silicate degree as was feasible.

At the same time the rate of concentration had to be kept carefully in mind for the two reasons which always obtain in such cases: first, to lighten the succeeding matte operations; second, because a low-grade matte meant that, instead of sending the iron into the slag and thus getting rid of it at once, it was being carried into the matte where it would have to undergo further treatment.

The results obtained by the older method were by no means unfavorable, according to the views then held of true pyrite smelting. In spite of the massive pyrite that constituted the principal portion of the charge, the ratio of concentration averaged 7 to 1; but this yielded a matte containing only about 22 per cent copper, and calling for a concentration smelting before it was fit for the converters.

The slag from the ore smelting is, however, the chief object of the present discussion. It was about a 4 to 3 silicate of the general type $3RO \cdot 2SiO_2$, being somewhat more basic than a bisilicate.[1]

This slag contained about 38 per cent silica with 48 per cent ferrous oxide, and had a formation temperature approximating 1070 deg. C.

It was recognized from the outset that this slag was too acid for conditions which demanded the fluxing of the maximum of FeO with the minimum of SiO_2, yet the metallurgist was helpless. The furnaces had been equipped with a certain set of conditions, and put in motion on this basis. This particular slag — which we will suppose to be 38 per cent SiO_2 and 48 per cent FeO — was the compound that corresponded to this particular set of conditions, and no feeding of additional pyrite would alter it. The excess pyrite simply melted and diluted the matte, and the

[1] See chapter on "Slags" for a study of any points about slags which may appear obscure in this illustration.

slag still continued to contain its 38 SiO_2 and 48 FeO. Similar conditions could only produce similar results.

The problem that confronted the Mount Lyell smelter was how to equip the furnace with a new set of conditions which should correspond to a normal slag of say 30 per cent SiO_2 and 55 per cent FeO — thus approaching a unisilicate, and having a silicate-degree of only about 1 to 1, or $2RO \cdot SiO_2$. The formation temperature of slags diminishes as the percentage of silica increases until quite a degree of acidity is reached — some 50 per cent in fact. Conversely, the formation temperature increases as the silica diminishes, and the desired 30 per cent SiO_2 slag has a formation temperature of 1170 deg. C., or about 100 deg. C. higher than the former 38 per cent silica slag.[1]

It followed, therefore, that if Mount Lyell could establish and maintain a temperature of about 100 deg. C. higher in the smelting-zone of its furnaces, conditions would be suitable for the formation of a slag corresponding to this increased temperature, and this slag (with the constituents under consideration) would be the desired compound of about 30 SiO_2 and 55 FeO.

From the nature of the process, coke could not be used to generate this increased heat; consequently it was necessary to burn more pyrite per minute. This meant a corresponding increase in the amount of oxygen blown into the tuyeres per minute and, as the ore-column then used could stand no heavier blast, it became necessary to heighten the furnaces. At the same time it was decided to drop the heating of the blast, experiments having convinced Sticht that, in true pyrite smelting, cold blast tended to more vigorous oxidation, higher temperature, and better concentration.

The results justified his expectations to the fullest degree.

[1] In studying the formation temperature of slags it is always disturbing to read in books that a 50 per cent silica slag — which we know from experience is likely to freeze the furnace promptly — has actually a lower formation temperature than a fiery 30 per cent silica slag, which runs as thin as water.

This is due to the fact that there is no definite relation between the *formation* temperature of a slag and the temperature at which it will *flow* freely and comfortably.

The *flowing* temperature is the only one which interests the strictly practical metallurgist; the *formation* temperature, on the other hand, has especial charms for the research department; but the best technical result can be obtained only by the man who is thoroughly at home with both, and knows how to take advantage, commercially, of this knowledge.

The temperature of the focus rose, the slag became more ferruginous, and the ratio of concentration doubled, the matte averaging about 40 per cent copper.[1]

Having disposed of the slag-forming constituents of our illustrative charge, we may now follow the small proportion of sulphides which has escaped the fierce oxidation of the bessemer-zone and has sunk into the hearth, where we may suppose that it undergoes no further change.

The only sulphides which we have to consider are the modified sulphides resulting from the exposure of the pyrite and chalcopyrite to the conditions which have just been studied. As already explained in detail, such portion of the pyrite as has escaped oxidation has become $Fe_n \cdot FeS$, so that there remains only the chalcopyrite to follow.

The behavior of this mineral when exposed to heat has been fully considered in Chapter IV, where it was explained that, under the influence of heat alone, it might be considered to become $Cu_2S \cdot FeS$. The behavior of the FeS portion is already understood and, as Cu_2S undergoes no further change, it is evident that the matte resulting from a pure cupriferous pyrite ore, when exposed to the true pyrite process, must consist of Cu_2S and $Fe_n \cdot FeS$, or FeS.

It is astonishing how little the copper of the Cu_2S is affected by the powerful oxidizing action of the bessemer-zone. So long as a fair amount of FeS accompanies it, the copper is protected, and the slags from this process are as free from oxidized copper as are equally ferruginous slags produced in the coke-burning furnace.

A small addition of coke is found beneficial, and even necessary, at most pyrite furnaces, even in those cases where the pyrite itself should be able to furnish more than enough heat for the entire operation. The exact function of this coke is still a matter of argument. It burns in the upper regions of the shaft by reducing SO_2 to S ($2C + SO_2 = 2CO + S$), and, consequently, can scarcely have much effect at the critical part of the operation; yet its omission is apt to be followed by chilled tuyeres, accretions, and similar signs of distress.

[1] It must be understood that this mention of the superiority of cold over hot blast refers solely to the conditions prevailing in true pyrite smelting. In partial pyrite smelting heated blast is frequently advantageous and occasionally indispensable.

At Mount Lyell, after the improvements to which I have referred had been effected, one-half to one per cent of coke sufficed to keep the furnace in good condition, although at present, with a lessened proportion of pyritous ore and with much lead and zinc in the Mount Lyell pyrite, two to three per cent of coke is required to effect the same purpose.

The main point to recollect in regard to the employment of coke in the true pyrite furnace is that it must not get down unconsumed so far in the shaft as to lessen the oxidizing effect of the blast. As soon as this occurs, quite a new set of conditions arises, and the process must begin to be classed as "partial pyrite" smelting.

Metallurgical writers occasionally refer to the smelting of pyrite ores at Tilt Cove, Newfoundland, without coke, as of something mysterious and unparalleled, and compare it with the work at Mount Lyell, Ducktown, Keswick, and other pyrite plants, much to the disadvantage of the latter. This is a misleading comparison, as the Tilt Cove process is operated with a different aim and with different results from the other furnaces enumerated. Its originator, T. D. Nicholls, of the Cape Copper Co., modified the pyrite process admirably to suit the peculiar requirements at Tilt Cove, but would be the last person to claim that he had effected results which were unattainable by the other metallurgists who were employing the more common type of the same general method.

A brief description of the actual practice at this isolated little plant will serve as a useful illustration, and may discourage further unjust comparisons.[1]

The Tilt Cove ore is a nearly massive pyrite having, approximately, the following composition:

Cu	3.7 per cent
S	35.0 " "
Fe	37.0 " "
SiO_2	13.0 " "
Al_2O_3	2.5 " "
CaO and MgO	1.0 " "
	92.2 " "

[1] I take many of my figures from an instructive paper by F. S. Nicholls, superintendent, entitled "Pyritic Smelting in Tilt Cove, Newfoundland." *Engineering and Mining Journal*, Sept. 5, 1908.

Up to the year 1895, the ore was roasted in lump form and smelted with coke in small brick blast-furnaces of about 50 long tons (50.9 m.t.) daily capacity. The use of expensive coke seemed unreasonable with an ore containing so much fuel in the shape of sulphides, and T. D. Nicholls undertook to adapt the plant to pyrite smelting, the first new furnace blowing-in on Jan. 1, 1895. It must be borne in mind that the pyrite process was then in its infancy, and that it could claim but little experience and still less literature.

The four small unjacketed furnaces were of the simplest description, and differed from the former coke-burning furnaces only in having eight instead of three tuyeres, and in being cut down so that there was only a 24-in. (0.61 m.) space between tuyeres and feed-plate. They were blown out every Sunday to give the men a respite from the excessive smoke.

To start a furnace, 500 lb. (227 kg.) of coke was kindled upon the hearth and, when burning brightly, was covered gradually with ore until the little shaft was filled. As soon as the ore was well in glow, a light blast was started and was soon increased to the normal pressure of 12 ounces. No silicious flux was used, the slag was very basic — carrying about 0.25 per cent copper — and the matte was extraordinarily low, averaging only seven per cent copper, and indicating a concentration of scarcely more than two into one.

The actual costs and results for one period of 294 days, as given by Mr. Nicholls, are of much interest, as they are obtained by the uncompromising method of taking from the books the total amount of money expended at mine and furnace during that period, and dividing it by the number of tons of ore smelted. During the period just mentioned 40,133 long tons of ore were smelted, having an average copper content of 3.41 per cent, while the matte contained 7.06 per cent copper. The total costs for mining, transportation of ore to smelter, smelting, and piling the matte for shipment — including labor and material — was 55,053 dollars, or $1.37 per long ton of ore.

Experiments were conducted at the home plant in South Wales in bessemerizing this matte, sand being blown into the converter along with the blast. No trouble was experienced in bringing it up to 50 per cent copper in a single operation.

It is not, of course, reasonable to compare this peculiar process

with ordinary true pyrite smelting, where a concentration of between 5 and 20 to one is effected. The unprecedentedly low ore-column utilizes but a fraction of the heat, and the light blast and low silica permits the combustion of but a fraction of the ore-fuel; yet this fraction of a fraction is sufficient to melt the charge thoroughly, and to keep furnace and settler in good condition.

The regular smelting operation was conducted absolutely without coke, none of this material being used excepting when emergencies arose due to abnormal amounts of gangue-rock coming down with the ore. Mr. Nicholls remarks that, under ordinary conditions, the furnace ran better without coke than with it.

This illustration scarcely demands discussion. The work, as carried out, solved a problem quickly and cheaply, and made an ore distinctly profitable which had been, hitherto, a somewhat doubtful asset.

If the Tilt Cove lenses had appeared large enough to warrant a heavy investment, Mr. Nicholls would doubtless have conducted his pyritic work with the high ore-column, heavy blast, and adequate silica to ensure a much higher degree of concentration — although, as the ratio of concentration increased, the necessity for the addition of a small proportion of coke would certainly have accompanied it.

Returning, after this digression, to the simple charge of quartz, pyrite, and chalcopyrite which we have been considering as a typical mixture for purposes of study, we notice that there yet remains unmentioned one striking point of difference between the pyrite process and the ordinary smelting of oxidized ores with coke. In the latter operation, all of the constituents of the slag which is about to be produced are present in the ore-charge as it is fed into the furnace. Silica and iron oxide are both there, ready to unite without delay or change (excepting the automatic reduction of Fe_2O_3 to FeO).

The coke, to be sure, demands oxygen from the outside to burn it; but the coke — except for its slight proportion of ash — has nothing to do with either the resulting slag or matte and cannot, therefore, be regarded as a part of the ore-mixture.

In the pyrite charge, we have the silica ready to fulfil its

office, but no iron oxides are present to combine with it; nor can there be any until we have introduced an entirely new substance into our ore-mixture. This new substance is, of course, oxygen and, although we are not in the habit of regarding it in this light, is still as real and essential and weighable a constituent of the mixture as is either the silica or the iron.

Not a pound of silica can be slagged until it is provided with a suitable amount of FeO, and not a pound of FeO can be produced until the iron sulphide is provided with a certain amount of oxygen.

Looking at the process a moment from this new standpoint, we might assume that we had a furnace through whose tuyeres were entering 10 lb. of oxygen per minute which required fluxing.

For this purpose, the 10 lb. of oxygen will require a certain specific amount of Fe and S, which it must obtain for itself by decomposing the FeS which we will furnish as required. Such FeO as is produced by the 10 lb. of oxygen will take up the precise amount of silica required to make a silicate having the formation temperature corresponding to the heat evolved by the union of the 10 lb. oxygen with the necessary amount of iron sulphide.

Finding that the furnace is running too slowly, we determine to increase its capacity, and we blow into the tuyeres twenty pounds per minute of oxygen instead of ten. This doubles the production of FeO which, consequently, takes up twice as much silica as before and (leaving out of sight the change in the composition of the slag which results from instensifying the combustion) the weight of slag produced per minute will be doubled.

Expressing the same idea in figures, we may assume our slag-forming constituents to be three in number instead of only two, as heretofore. Up to this point, we have spoken of them as consisting only of ferrous oxide and silica; but as the ferrous oxide is not yet formed and only exists potentially, in the shape of FeS; and as the FeS is merely a secondary product derived from the pyrite that was fed into the top of the furnace, it will be more convenient, for the moment, to express the ferrous oxide in terms of ferric sulphide, or pyrite. The three slag-forming constituents on this new basis will be oxygen, silica, and a portion of the pyrite (the other portion being volatile).[1]

[1] In order to simplify the argument, I omit any consideration of the small amount of Fe resulting from dissociation of the FeS before oxidation

As the pyrite is the only substance in the mixture that acts as a flux for the oxygen, it would seem essential to know how much pyrite one pound of oxygen will burn. This method of calculation, however, would be incorrect, as the oxygen does not burn all of the pyrite; it only burns the residue that remains from the pyrite after one atom of its sulphur has been sublimated by heat alone. This substance may be assumed, for the moment, to be FeS, and we require to know what weight of FeS will be burned to FeO and SO_2 by one pound of oxygen.

$$FeS + 3O = FeO + SO_2$$
$$88 + 48 = 72 + 64$$

Thus 48 lb. O burns 88 lb. FeS

1 lb. O burns $\frac{88}{48} = 1.833$ lb. FeS

Incidentally we shall need to express the FeS in terms of the FeS_2 from which it was derived:

$$FeS + S = FeS_2$$
$$88 + 32 = 120$$

Thus 88 lb. FeS was derived from 120 lb. FeS_2

1 lb. FeS was derived from $\frac{120}{88} = 1.364$ lb. FeS_2

or 1 lb. FeS_2 produces $\frac{88}{120}$ $= 0.733$ lb. FeS

The 10 lb. oxygen, therefore, burns 18.33 lb. of FeS (or 25 lb. of FeS_2), and forms therefrom $\frac{72}{88} \times 10 = 8.18$ lb. FeO, which requires 3.4 lb. SiO_2 to form a unisilicate.

Thus the total fluxes for the 10 lb. of oxygen will consist of

silica	3.4 lb.
pyrite	25.0 lb.

Or, putting the statement in more convenient form (recollecting also that it is based upon assumptions that require corrections): for each pound of oxygen that is blown into the tuyeres, there must be supplied 0.34 lb. silica and 2.5 lb. pyrite.

It may appear fanciful to talk of oxygen as a slag-forming constituent of the charge, and to select it as the leading substance which must be fluxed by the addition of certain exact amounts

begins. I also base the calculation on the complete utilization of the oxygen of the blast, although, in actual practice, 70 per cent of the same would be nearer the truth. Moreover, I neglect the small amount of FeS that escapes oxidation, enters the matte, and is thus removed from our present sphere of inquiry.

of silica and pyrite; and, of course, such a peculiar method of calculation would never be employed in actual practice.

The object of the illustration, however, is to focus the attention of the metallurgist upon the fact that — other things being reasonable — it is the volume of blast at his disposition that determines the capacity of his furnace and the intensity of his combustion; and if more metallurgists had begun their slag calculations at the blast-end of the plant instead of at the ore-end, there would have been fewer failures in pyrite smelting.

Next to ore-supply — and, indeed, of equal importance for successful smelting — is the question of wind-supply. The net capacity of blowers is nearly always overestimated, and the requirements of the pyrite furnace are equally often underestimated.

Assuming for the air an efficiency of 70 per cent, it will take about 80 cu. ft. of air to furnish one pound of oxygen; and to oxidize one pound of FeS_2 to $FeO + SO_2 + S$, there will be required 32 cu. ft. of air.

Knowing the number of pounds of pyrite per minute which experience shows should be oxidized for a hearth of given size, we have at once a basis of calculation for the blowing-plant. Assume that we expect to feed into the furnace, in addition to silicious flux and the gangue-rock of the pyrite, 250 tons of solid pyrite per 24 hours, and to decompose and oxidize (on the basis just explained) 88 per cent of this pyrite. How much air per minute will be required to accomplish this?

As the volume of air is expressed most conveniently in cu. ft. per minute, the duty of the furnace should be reduced to the same unit of time. Eighty-eight per cent of 250 tons per day is about 305 lb. per minute. One pound of pyrite requires 32 cu. ft. of air, and 305 lb. will require 9660 cu. ft. Corrections for elevation should be applied to this result, and, in spite of the fact that we have allowed only a 70 per cent efficiency for the air, a blowing capacity of at least 12,000 cu. ft. per minute should be provided, as diluted air reacts but feebly with the sulphides.

Having considered two of the three main slag-forming constituents — the oxygen of the blast and the pyrite with its potential content of FeO — we may turn, finally, to the silica.

In true pyrite smelting, the main aim and object of the metal-

lurgist is to rid his ore of its *iron*. This statement becomes the more obvious as the purity of the pyrite increases; and as any proposition is more clearly represented by beginning its study with a typical and extreme instance, I will assume, for the moment, that our ore consists solely of a cupriferous pyrite which is absolutely massive and devoid of gangue-rock.

Setting aside the small fraction of the constituents of this ore that we desire to retain — the Cu_2S, with a little diluting FeS — we have nothing remaining but pyrite. Our object is to get rid of this pyrite, and we know that its component parts, Fe and S, will go in two different directions: the S up chimney, and the Fe into a suitable slag, provided we offer it the necessary inducements.

Next to keeping up an ample supply of oxygen, the main inducement is to provide an eager acid at the moment when the FeS is ready to form FeO, and yet we cannot do this unless some free strong acid is in contact with the forming FeO, to combine with it and hold it where it is.

The only acid available for this purpose is free silica: not silica already in combination with lime or magnesia or alumina or other iron, as is so often the case in silicious ores, as well as in the gangue-rock of the pyrite itself.

This is another of the vital points in pyrite smelting, and its non-recognition has cost several great companies more dearly than any but the experts know.

The pyrite furnace may digest a reasonable amount of such silicates as I have just enumerated, but its attitude toward them resembles closely the attitude of the coke-burning furnace toward its inert and helpless charge. It treats them simply as so many pounds of indifferent and lifeless material which absorbs heat without giving any sort of aid in return, and which, by the continued application of such spare heat as it possesses, may be melted, and even driven into taking up a little FeO to make itself more fusible.

Even the most acid of these silicates offers no inducement to the iron to quit its protecting sulphur. This demands a vigorous exacting acid; and when the sharp edge of the silica has been blunted and its characteristic qualities impaired by any previous base, it is of no value to the pyrite smelter in furthering the reaction which is the main object of his efforts.

Granite, slate, schist, porphyry, feldspar, mica, garnet, and the numerous other rocks and minerals that so often form a portion of our furnace-mixtures may all be smelted in the pyrite furnace to an extent commensurate with such superfluous heat as the oxidation of the active portion of the charge may have evolved; but let no one regard their content in silica as of any avail in supplying the acid needed in the decomposition of the pyrite. Their silica is already pre-empted, and plays no active part in the process.

It follows, therefore, that the silica needed for this latter purpose must be *free silica:* such as quartz, quartzite, sandstone, and the like; that its quantity must depend upon the amount of FeO formed; and that this factor again is governed by the weight of oxygen blown in through the tuyeres; so that by whatever path we travel, we always come back eventually to the one central governing influence of the whole operation — the blast.

A clear appreciation of these fundamental principles should enable any one to determine the suitability of the true pyrite process to any given set of conditions, so far as theoretical knowledge is concerned.

The technical management of the operation itself is quite a different matter. It is difficult, variable, subject to periods of decline without apparent cause, and extremely slow in recovering its normal condition. This results from the fact that, as no addition of coke can be made without affecting profoundly the degree of oxidation and thus upsetting completely the ratio between base and acid, the ailments of the furnace cannot be mitigated by outside aid, and it is slow and hard work for a furnace already below par to keep up the reactions necessary for its mere existence, let alone providing a surplus of restorative energy.

Neither in size nor in shape do the blast-furnaces in use for true pyrite smelting differ particularly from the ordinary copper blast-furnaces, except that with heavier blast and faster driving, their height is becoming considerably greater than would be suitable for a roasted charge. This statement as to size and shape is, however, applicable to the mere skeleton of the furnace as provided by the metallurgist. The process itself modifies profoundly the conditions within the shaft, and constructs auto-

matically — out of the infusible constituents of the charge — an inner contour suited to its own requirements. While this contour might be imitated by the builder at the outset, it varies so much, as these requirements change, that it is impracticable to construct the interior in any way to conform to the contour which the process endeavors to establish.

The most characteristic feature of a furnace doing true pyrite smelting is a light porous mass of accretions in the active zone bridging the shaft more or less completely across, and having its greatest development at a point somewhat above the tuyeres. Below this section, the accretions disappear rapidly into the cavity of the hearth, whilst above they thin out more slowly, forming an artificial bosh.

In a rectangular furnace, an ideal section across the horizon of greatest development would show the shaft narrowed down to a mere slit — called by Sticht the bessemer-slit — and at points even bridged across so completely that a bar may sometimes be driven into a tuyere, and out on the opposite side, without encountering slag, or even heat.[1]

This condition, so alarming and critical to the ordinary smelter, is essential to the normal action of the pyrite process. Bessemerizing is a highly vigorous and rapid process, and demands intense concentration and localization. The inert quartz boshes act simply as a support to the charge above, and are more or less stationary and permanent. The bessemer-slit, on the contrary, is the seat of intense activity and, although filled with the column of quartz fragments which represent the residue of the ore after liquation has taken place, is in a condition of motion — the column descending from above as fast as it is fluxed away below.

The numberless channels and interstices in this descending column of white-hot silica-fragments offer a passage to the ascending blast as well as to the descending rain of sulphides and, at the same time, provide a very large surface for the attack of the resulting FeO.

If the furnace is receiving too much silica, the permanent accretions will continue to grow until the process is blocked; if

[1] See a paper by Freeland on this subject in connection with the Ducktown furnaces, in the *Engineering and Mining Journal* of May 2, 1903. The accompanying illustrations, showing ideal vertical and horizontal sections of these furnaces, are taken from this paper.

too little, the matte will at once become diluted with the FeS which has failed to find sufficient inducement to become dissociated; or, if actually driven to oxidation by an excess of blast, will burn to the sticky half-melted higher oxides of iron, and kill the process.

That this porous, friable mass of quartz is not a half-imaginary object founded upon such unsatisfactory observations as those made while the furnace is still in operation, is shown by Sticht's observations.[1]

He states that formerly, at Mount Lyell, this dead layer of porous quartzose material that formed the lateral boundaries of the bessemer-slit remained more or less intact after a furnace was blown out, and thus lent itself to thorough investigation. He says, "it was composed of quartz fragments and of slag — never of Fe_2O_3, Fe_3O_4, nor of matte."

The complete liquation of the sulphide from its accompanying quartz at a point comparatively high up in the shaft is a fact that does not require this proof for any one familiar with the melting-point of FeS; but the complete absence of any Fe_2O_3 or Fe_3O_4 will be of interest to those who neglect all consideration of the heat-balance of the pyrite opera-

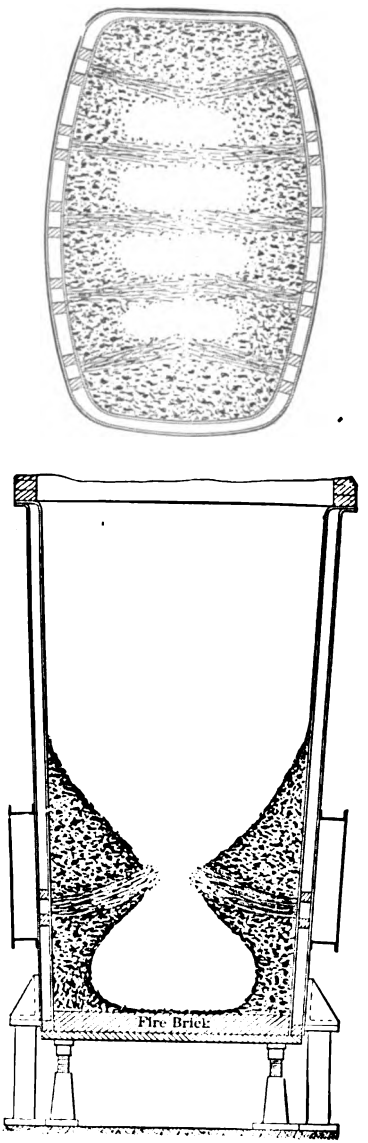


FIG. 49. — Ideal vertical and hor. cross-sections of pyrite furnace

[1] See Sticht's paper entitled "Ueber das Wesen des Pyrit-Verfahrens," *Metallurgie*, 1906.

tion, and regard the process as a combination of roasting in the upper part of the ore-column and smelting in the lower portion.

Since the substitution of cold blast and more active driving at Mount Lyell, these artificial boshes do not survive the blowing-out, although they are very much in evidence during the normal running of the furnace. Indeed, with Sticht at Mount Lyell, as with Freeland at Ducktown, the tuyeres are always dark, and a bar may often be driven clear across the furnace without encountering fire. They must, of course, be kept sufficiently open to permit the free entrance of the blast; but normally they show no fire.

All this means simply that the zone of oxidation begins at some little distance above the tuyeres, and is exceedingly circumscribed and concentrated.

The upward extension of this zone of oxidation is limited by the volume of the blast and the extent of contact-surface between the air and the descending streams of sulphide. If the drops of molten sulphide were sufficiently minute, and abundant air were provided, complete oxidation would be practically instantaneous, as is the case in dropping fine particles of sulphides through heated oxidizing gases, or in burning pulverized coal.

As this is not the condition in the pyrite furnace, the oxidation of the melted globule can proceed only upon its surface, and, although practically instantaneous so far as regards the surface with which it is in contact, yet requires a perceptible period of time before it can work its way — layer by layer — to the center of the globule. This requires an appreciable time, and this space of time is afforded by increasing the length of the contact between air and sulphide. Consequently, an increase in the blast means an increase in the height of the bessemerizing zone, and a corresponding decrease in the upper preheating, dissociating, liquating horizon.

For the purpose of conserving the heat already evolved, as well as for providing the time necessary for the various preparatory stages, the volume of the preheating zone must stand in a tolerably close relation to that of the focus, and it is impossible to enlarge the latter at the expense of the former without serious trouble and loss. Consequently, in cases where a furnace has already attained its full capacity, the steps involved in increasing

its duty would be at least three in number: first, to increase the blast; this would enlarge upward the limits of the focus, so that it would be necessary, second, to increase the height of the shaft; the augmented blast and greater area of focus would ensure a much more extensive and rapid oxidation of the sulphides, so that, third, it would be necessary to add more silica. It is evident that these three factors stand in close relation to each other.

This plan of regarding oxygen, pyrite, and silica as the three vital and mutually interdependent constituents of the true pyrite smelting process provides some sort of basis on which to plan the prospective work. No such nicety of result can be expected as in ordinary coke smelting, where the various component parts of the mixture are already prepared, and only await the addition of heat to rearrange themselves into new substances, the weight and composition of which may be determined in the laboratory with remarkable accuracy. The whole will of this process is subordinated by the coke, and can be controlled accordingly; but the pyrite furnace has to have a free hand to carry out its elaborate reactions and, until it has shown what it intends to do, the metallurgist must follow rather than lead.

It is only after it has established its normal temperature and slag, and shown what it intends to do under the prevailing conditions, that the metallurgist can attempt to carry out his own wishes with reasonable accuracy and certainty; and this he effects not by futile endeavors to force the process in a direction contrary to natural laws, but rather by establishing the conditions essential to the results that he desires to obtain. In almost every case, the change of conditions is brought about solely by varying the relation to each other of the three factors just discussed.

I have devoted much space to this matter of blast, because it is the most important and most underestimated item in connection with the process. A new plant should not only be equipped with blowing capacity far beyond its theoretical requirements (even after figuring the efficiency of the air at only 70 per cent), but should be constructed in such a manner that both blast and furnace-height could be largely increased without too much extensive reconstruction. Care should be taken to provide ample draft and ventilation, owing to the enormous evolution of SO_2 and other offensive gases.

One word more may be said before leaving the subject of

blast. The amount of oxygen that you have at your disposition is not determined by the piston-displacement of the blower nor by the wind-pressure in the blast-pipe. It is determined by the amount of air that enters the furnace, and this amount bears no definite relation to the factors just mentioned. Probably this is the most important practical statement in this book, and should always be borne in mind by the inexperienced.

There remains yet one more important matter to consider while speaking of the pyrite charge: namely, the indifferent non-active substances which natural, or commercial, contingencies force us to accept as a part of the mixture that we are called upon to smelt.

Amongst these substances it is not proper to place the free silica required by the prospective FeO to form the slag corresponding to the temperature which is established by the combustion of the pyrite. Indeed, this silica is not only an essential but an active constituent as, in addition to its powerful aid in the dissociation of FeS, its union with this base develops an important fraction of our somewhat scanty supply of heat.

These indifferent substances which are, practically, a dead weight to the process vary greatly with local conditions, but may commonly be grouped under some of the following heads: gangue-rock of the massive sulphide ores; gangue of other ores which are added to the mixture for commercial reasons; limestone flux (seldom used in this highly basic process except to lighten the slag); ballast-slag, accretions, sweepings, etc. If extreme accuracy is desired, we may also include the trifling quantity of ash from the small addition of coke, and also the foreign substances carried by the fluxing-silica.

The essential feature of all of these substances — which may be designated as "earthy materials" to distinguish them from the quartz and pyrite of the mixture — is that such silica as they contain, apart from their content in free silica which has already been allowed for, is in a state of combination, and therefore unavailable for active duty.

Apart from losing moisture, carbon dioxide, etc., all of these substances are lifeless, both chemically and physically, until the active constituents of the charge have supplied the heat required to soften them and awake such feeble chemical affinities as they

may still retain. In other words, they undergo no change until they have sunk to a level in the shaft where the heat begins to cause some degree of melting. Even here they undergo scarcely any chemical change, because they consist largely of lime, alumina, and magnesia silicates — or of calcium oxide from the limestone — and none of these substances are really fusible at any temperature that they are likely to encounter short of the focus.

It is for this reason that the silicious skeleton filling the shaft of the furnace is so porous, friable, and easy to excavate when the furnace is blown out. There is scarcely any formation of slag in these upper regions and, consequently, nothing to hold it together. The sulphides have liquated away from it, leaving behind the infusible skeleton, and this infusible quartz together with the refractory natural silicates just mentioned are both awaiting — the former with avidity, the latter with indifference — the one indispensable base which alone has the power to convert them all into fusible silicates. This base does not exist in the upper regions of the shaft. It is manufactured to order in the focus, and when this region is at length attained, base-forming, heat-forming, and slag-forming all proceed vigorously and simultaneously.

Bearing this in mind, it becomes evident at once that the proportion of earthy or indifferent or non-heat-producing substances that the pyrite furnace can carry depends entirely upon how much heat it has to spare after attending to its own indispensable constituents — pyrite ore and silica.

This point might be determined theoretically for each individual case by careful thermal calculations, but our knowledge and experience are yet too imperfect to enable us to base actual plans upon calculation of this nature, save in a very general way.

Beginning with the other end of the problem, we may glance at a few of the cases where we know positively that sulphide ores have been melted thoroughly by the heat of their own combustion.

For instance, in his original experiments in smelting the massive Spanish pyrites in a bessemer converter, Hollway produced a 50 per cent matte from ores carrying less than two per cent of copper, and still had a large excess of heat.

The regular work at Tilt Cove, Newfoundland — already

described in this chapter — melted down tolerably massive pyrite under conditions which effected a concentration of only two to one. In spite of this very feeble degree of oxidation, and of the enormous loss of heat in the low furnaces employed, there was abundant heat left for the purpose.

Wright, at Keswick, smelted for months at a time a charge composed of

cupriferous pyrite	74.3	per	cent
flue-dust	9.0	"	"
slag	2.8	"	"
fluxes	13.9	"	"
	100.0	"	"

copper in matte, 26.8 per cent; rate of running, 8.75 tons per sq. ft. hearth area (85.4 m. t. per sq. m.). Coke used, 0.32 per cent of total burden.

This mixture was smelted for long periods at a time without any coke at all, but it was found that the addition of the small proportion noted had a strong effect in lessening accretions and in smoothing out irregularities. In this particular instance a moderately heated blast was used, and Wright considers that this had an important effect upon the running of the furnace.

Sticht, at Mount Lyell, using a mixture rather high in pyrite, though still loaded with a considerable amount of fluxes and ballast-slag, with cold wind and a ratio of concentration of 18 or 20 to 1, could run for several days together without coke, but found that the furnace had a tendency to become cold and abnormally sensitive, and that this could be corrected by the use of one-half to one per cent of the weight of the entire charge in coke. This small proportion was consumed completely in the upper, non-oxidizing region of the shaft, obtaining the oxygen necessary for its combustion by reducing SO_2 to S and, consequently, had no direct effect upon the normal temperature of the focus.

He estimates that this coke furnished about 68 calories per unit of charge, or about one-eighth of the total heat development, and points out that a very moderate increase in furnace activity would replace it.

Basing his calculation upon analyses of solids and gases, and upon actual determinations, Sticht offers the following figures in

connection with the Mount Lyell pyrite smelting when using 1.1 per cent of coke.[1]

Sticht estimates a total net heat development of 654 calories per unit of charge. He distributes the heat consumption as follows:

Waste gases and flue-dust	106.4 Calories	=	16.2 per cent of total heat
For the newly formed slag	185.5 Calories	=	28.4 per cent of total heat
For the ballast-slag	33.8 Calories	=	5.2 per cent of total heat
Heat for total slag	219.3 Calories	=	33.6 per cent of total heat
For the matte	8.7 Calories	=	1.3 per cent of total heat
Total for slag and matte	228.0 Calories	=	34.9 per cent of total heat
Evaporation of moisture	12.4 Calories	=	1.9 per cent of total heat
Dissociation of CO_2	22.1 Calories	=	3.4 per cent of total heat
Sublimation of S	17.2 Calories	=	2.6 per cent of total heat
Decomposition of Fe sulphides from FeS_2 to Fe_5S_4, including latter	177.9 Calories	=	27.2 per cent of total heat
Jacket-water	78.7 Calories	=	12.0 per cent of total heat
Radiation, etc., and losses, by difference	11.3 Calories	=	1.8 per cent of total heat
Total	654.0 Calories	=	100.0 per cent of total heat

These figures are based upon analyses and actual determinations.

A summary of the useful work accomplished by the total heat development shows that, in spite of the heavy loss in the furnace gases, the actual duty of the pyrite furnace is comparatively high.

The chemical reactions and the fusion require:

For the molten products	228 Calories	=	34.9 per cent of total heat
For dissociation and decomposition	229.6 Calories	=	35.1 per cent of total heat
	457.6 Calories	=	70.0 per cent of total heat
Losses	196.4 Calories	=	30.0 per cent of total heat
Total	654.0 Calories	=	100.0 per cent of total heat

These results compare favorably with calculations made at various coke-burning copper plants, and indicate a reasonable efficiency for the pyrite process. Still, so far as the development

[1] I can give here only the net results of these calculations. For the detailed assumptions and methods, see Sticht's essay in *Metallurgie*, already cited, or "Principles of Copper Smelting," pp. 272 to 283.

and utilization of heat is concerned, it is at a distinct disadvantage in having to waste such a large quantity of heat in dissociating the iron sulphides and volatilizing the sulphur. On the other hand, the coke furnace has to reduce large quantities of the higher iron oxides to FeO, and is also unable to apply the heat derived from the combustion of coke in so advantageous a manner as the powerful exothermic slag-forming reaction of the pyrite process.

Sticht dwells strongly upon the advantages to be gained by heavier blast (cold), and higher furnaces, thus increasing the actual intensity of the process as well as utilizing more thoroughly the heat thus evolved. The late increase in height and blast at Mount Lyell has not only benefited the process there, but indicates that even the present eighteen-foot (5.5 m.) ore-column may be increased with advantage.

While the advantages of drying the blast do not appear to be likely to benefit the copper blast-furnace process to the same extent as the iron furnace, it may well be that, where the heat-balance is so close as in the pyrite process, the advantage thus gained might be sufficient to remove many of the slight irregularities and troubles which arise from working with so narrow a margin.

The process at Mount Lyell, Tasmania, as the oldest and most distinguished commercial application of true pyrite smelting, will serve as the most fitting illustration of the technical workings of this method, and is the more suitable because, beginning with a strong predilection for preheated blast, it learned by trial and experience how to attain far better results without it. We also have abundant reliable information as to the work at this plant, all costs and results having been tabulated with accuracy for many years.[1]

The six blast-furnaces forming the first Mount Lyell smelting plant were 40 by 168 in. at the tuyeres, and $9\frac{1}{2}$ ft. high from tuyeres to edge of downcast. They had 32 three-inch tuyeres, with the

[1] I avail myself freely of the following articles, as well as of many private letters, by R. S. Sticht, general manager of the Mount Lyell Mining and Railway Company, "Ueber das Wesen des Pyrit-Schmelzverfahrens" *Metallurgie*, 1906. "Mining and Smelting at Mount Lyell, Tasmania," *Mineral Industry*, Vol. XVI. "Progress in Rapid Oxidation Processes Applied to Copper Smelting," address before the Australasian Association for the Advancement of Science, Adelaide, Jan., 1907.

exception of No. 3 furnace, which was used only for concentration of matte, and which had only 24 tuyeres, and a hearth area of only 36 by 126 inches.[1]

The five furnaces of No. 2 plant were erected in 1898 and were considerably larger, being 42 by 210 in. at the tuyeres, and 15 in. larger all around at the throat. They had 48 three-inch tuyeres, and their ore-column — originally the same as that of the furnaces of No. 1 plant — was increased to 11 ft. (now increased to about 18 ft.). The accompanying illustrations show this latest model. There is a special mechanical push-arrangement for charging which has proved satisfactory, and the model is especially interesting as consisting entirely of cast-iron jackets.

The steel-plate jackets of the first furnaces suffering seriously from corrosion by the cooling water, recourse was had to cast-iron jackets. These were gradually modified into solid cast-iron pipe-cooled jackets, the water-space being omitted excepting in the tier surrounding the hearth which seldom requires renewal. As will be seen in the illustrations, this arrangement has again been slightly modified in the newest furnace, which has four tiers of jackets, extending from hearth plate to downcast.

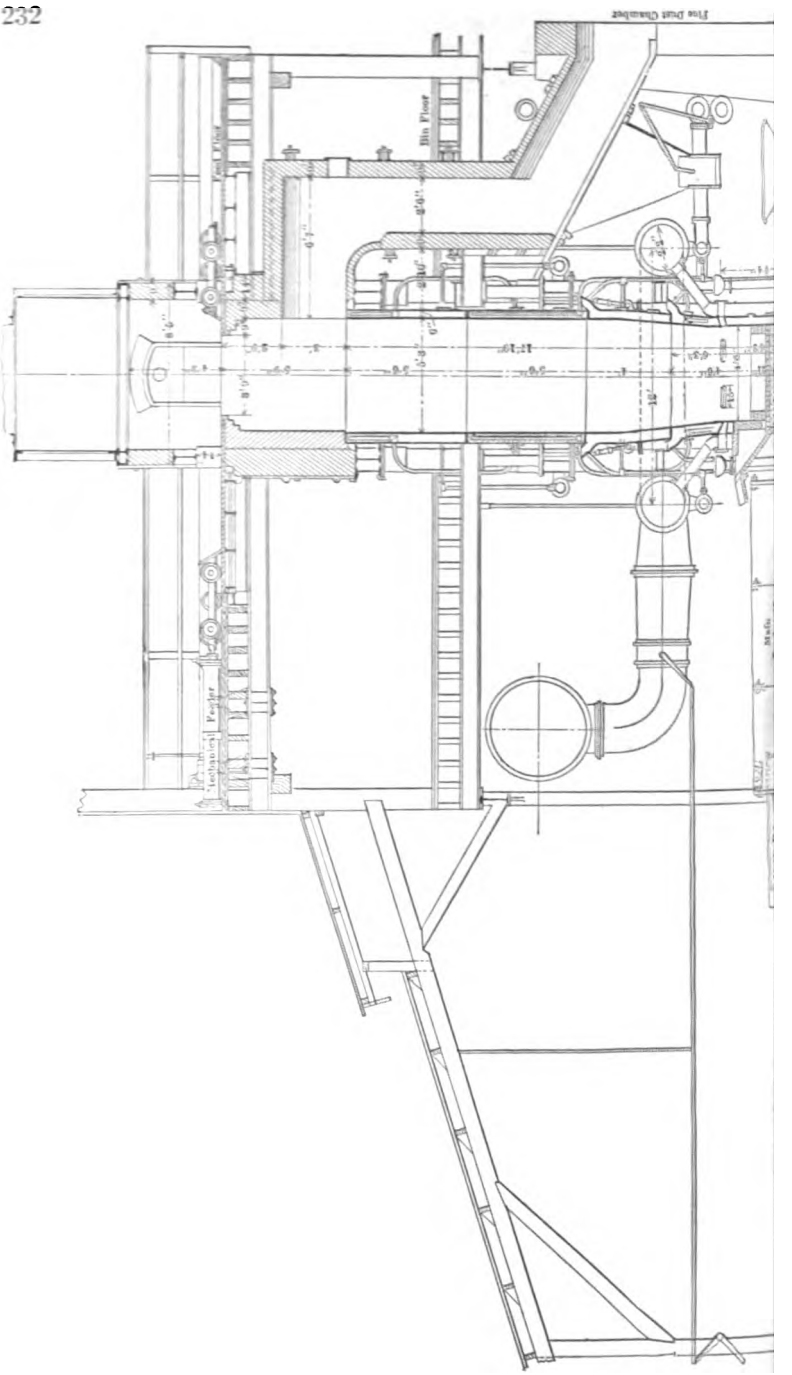
The hearth is 30 in. deep below the tuyeres, and the trap-spout and breast-jacket are of pipe-cooled cast-iron.

The main forehearth is 5½ by 19 ft., and 30 in. deep. It is water-cooled by means of pipe-coil side-plates, and is tapped through a small solid cast-iron pipe-cooled tap-jacket into ordinary matte pots, the matte being cooled and remelted for the converters. Owing to the heavy ferruginous slag, it is found advantageous to employ several small auxiliary settlers.

The rate of concentration is extremely high, the matte-fall irregular, the grade of ore variable, and the sensitive process extremely liable to sudden eccentricities. For these reasons it is essential to watch the grade of the matte constantly, and the shift bosses are taught to make a quick copper determination by the Parkes cyanide method, which they accomplish, from a rod-sample, within a few minutes, and with a limit of error not exceeding one per cent. The work is carried on with eight-hour shifts, common labor receiving $1.85; tappers, $2.10; furnace-men, in charge of two furnaces, $2.50.

The furnaces were fed by hand, owing to the delicate nature

[1] One foot (12) inches = 0.305 m.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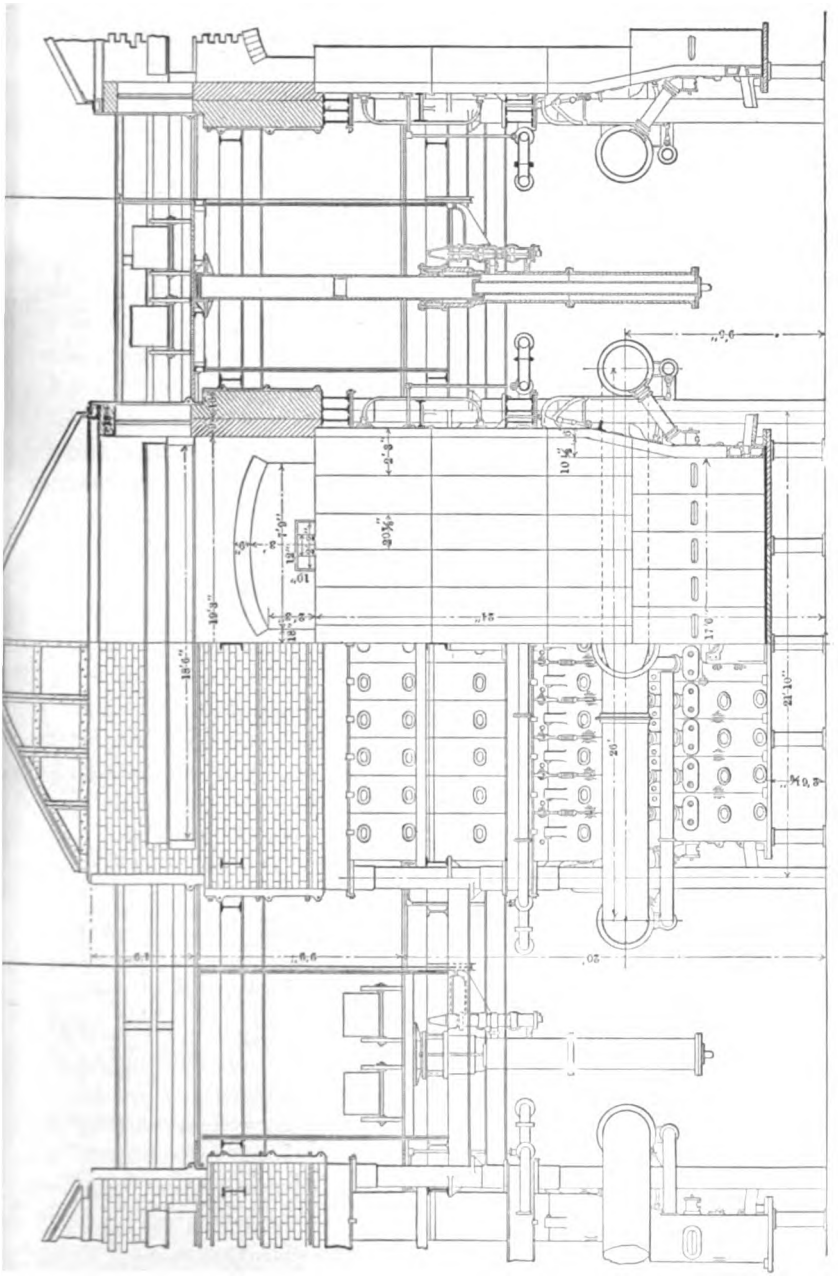


FIG. 51. — Remodeled Mount Lyell pyrite furnace — elevation

of the process; but, since 1906, there has been substituted a locally-evolved, patented feeding-appliance, which pushes the charges bodily off the charge-floor directly into the furnace. (See illustration.) The apparatus is hydraulically actuated, and consists of a steel frame provided with a line of hinged pushing-plates in front. A certain amount of skilled attention must still be given to the feeding, but this has been reduced to one feeder and a boy to each furnace.

The slag is granulated, the stream from the last settler being simply struck by a jet of water from a one-inch pipe, at 60 lb. pressure, and dropping into a launder in which the entire jacket-water of the furnace is united. There being insufficient fall for dumping purposes, within suitable distance, the slag-laden water is raised 24 ft. in height, and forced through a 10-in. level pipe over distances up to 500 ft. by means of a specially-designed motor-driven diffusor-chamber centrifugal pump. This makes 650 revolutions per minute, and is able to handle 1000 tons slag, with 16,000 tons water, with a brake horse-power of 40 to 45. The cost of this slag-elevation, including power, attendance, piping, repairs, duplicates, etc., is about two cents per ton of ore treated.

The adoption, long and careful trial, and final abandonment of a heated blast forms, perhaps, the most valuable and instructive feature of all of the admirable and original work which has been executed at this plant. It is, therefore, worth while to examine this little episode of metallurgical history with some care, as it is likely to repeat itself more than once in other parts of the world.

The hot-blast stoves in use at Mount Lyell were of the hanging U-tube type, substantially built, above customary capacity, and arranged on the counter-current principle. At the beginning, temperatures of 340 deg. C. were attempted, but were lowered gradually to 120 deg. C., to lessen the high consumption of fuel and because they were not found to be necessary.

The efficiency of the stoves at the higher temperatures was only about 50 per cent; but when the blast temperature was lowered, and the firing was reduced to such an extent that only about one-third of their grate area was used, it rose to 80 per cent. The construction was solid, and the apparatus was free from leaks.

Cautious experiments indicating that better results could be obtained with cold blast and that, if carbonaceous fuel was to be used, it could be applied more economically within the furnace, the preheating of the air was given up entirely, and the total blast-capacity of the plant, formerly suitable for from six to eight furnaces, was now used exclusively for the four furnaces of No. 2 plant. About 2.7 tons air per ton of ore-bearing material smelted is now used.[1]

With hot blast, the ratio of concentration was about seven to one. With cold blast, and on ore of about the same grade in copper, it was for the year ending Sept. 30, 1907, about 20 into one, figured on the mineral-bearing substances, the actual results being an average of 44.3 per cent copper in the matte resulting from a 2.25 per cent copper ore. This is achieved in a single smelting, so that the former matte concentration process was dropped.[2]

The thermal results of the true pyrite process are satisfactory when compared with the ordinary blast-furnace operations. As has been pointed out by various authors, the true pyrite process is severely handicapped by figuring all of the sulphur content of its ores as fuel when, as a matter of fact, less than half of it (even apart from the portion that enters the matte) is utilized, the other moiety being dissociated and volatilized, with loss of heat.

However, even with this serious loss and with the drawback of too hot waste-gases, due to insufficient height of ore-column,

[1] This comparison of the hot-blast and cold-blast processes at Mount Lyell is discussed so minutely in Chapter VIII of my "Principles of Copper Smelting," that I must confine myself here to a brief statement of results.

[2] In "Pyrite Smelting," page 261, J. Parke Channing, president of the Tennessee Copper Company, in describing the operations of the company's pyrite smelter at Ducktown, Tennessee, says: "It is almost unnecessary to state that in all of our work we have used nothing but cold blast; and thus far the use of hot blast in treating a heavy sulphide ore like ours (pyrrhotite) has not, as far as I can see, been of any advantage. At Mount Lyell it was formerly supposed that it was necessary; but, after four years' practice, the use of it has been finally abandoned. At the plant of our neighbors, the Ducktown Sulphur, Copper, and Iron Company, Mr. Freeland put in a U-pipe hot-blast stove, but could find no improvement either metallurgically or commercially therefrom, either on roasted or unroasted ore. He, therefore, for the present has abandoned any further attempt to utilize hot blast."

In partial pyrite smelting, however, conditions arise occasionally where the use of a moderately heated blast is accompanied by the happiest results.

the showing at Mount Lyell compares favorably with the ordinary processes which derive their heat from carbonaceous fuel. Sticht's investigations show that about 35 per cent of the total heat generated at Mount Lyell is devoted to the chemical work of smelting, and the fusion of the solid products. Another 35 per cent is absorbed by the dissociations preceding or accompanying the chemical reactions, and an abductive heat loss of 30 per cent, from various causes, makes up the balance. This direct efficiency of 70 per cent represents the degree to which the heat actually evolved during the smelting does the work which is expected of it. But owing to the fact that, as already noted, an important fraction of the sulphur present cannot act as fuel, the absolute efficiency, calculated on what the theoretical thermal capacity of FeS_2 might accomplish, is only about 60 per cent. Even this exceeds considerably the efficiency of ordinary copper-matting operations.

A minimum of 95 per cent of the iron of the sulphides is oxidized and slagged while, simultaneously, only about 35 per cent of the total sulphur is burned, the remainder escaping unburned within the furnace, or going into the matte.

The proportion of coke used at Mount Lyell has varied from nothing up to some two and one-half per cent. During the years 1904 to 1907, the coke consumption was below two per cent, taking into account the entire amount of this fuel used for blowing-in, drying, and other miscellaneous purposes, and including also a large amount of fines and 10 per cent of moisture.[1]

During the same period, the composition of the average slag was:

SiO_2	32.47	per cent
FeO	52.15	" "
Al_2O_3	7.22	" "
CaO	4.77	" "
BaO	0.90	" "
S	0.88	" "
Cu	0.39	" "
	98.78	" "

The average blast pressure was 47 ounces. The ratio of

[1] At present, the coke consumption is somewhat larger, owing to the use of a larger proportion of non-pyritous ores, and the appearance of much galena and blende in the deeper workings of the Mount Lyell ore-body.

concentration, figured only on the ore, was 17.4 to one. The assay of the matte in copper was 40.57 per cent.

The general recoveries of values in smelting and bessemerizing together from the beginning of the enterprise in 1895 to the end of Sept., 1907, were: copper, 85.72 per cent; silver, 92.57; gold, 104.28. These figures are obtained from careful records, accurate sampling, and close assaying, and by debiting the smallest traces of metals, and cover a period of 11 years, and a total burden of over four million tons of metal-bearing materials.[1]

The concluding portion of Sticht's presidential address at the Adelaide meeting of the Australasian Association for the Advancement of Science is so apt and enlightening that I quote a few of the paragraphs that are most appropriate to the present subject.

"The process (true pyrite smelting) as at present carried out is, however, not as perfect as it might be, chiefly because the full energy of reaction which pyrites and silica in proper proportions are capable of is not yet excited by the methods and appliances employed. The main defect is the necessity of still using at least a modicum of carbon. Complete emancipation from the introduction of carbonaceous fuel is, of course, the ideal; but even the best existing practice still finds necessary the application either of a hot blast, with or without coke, or of about one per cent to one and one-half per cent of coke (figured on the weight of furnace-charge) when cold blast is used. One reason is that, under present conditions of furnace height, there is too great a loss of heat at the furnace top — not so much through the non-combustion of the volatile portion of the sulphur as because the low columns used allow the gases to escape with too high a temperature.

"Wholly against a prevailing notion, the low column of the converter vessel should not be imitated. It is only an incidental feature of that vessel. Compliance with the true bessemer principle first of all requires complete absorption of the oxygen blown in — therefore, adequate height of column. The gases should leave the furnace free from uncombined oxygen, and the charge should be kept as high as at all compatible with the ready descent of the stock. The height of the stock is, consequently, a func-

[1] The plus-recovery of gold is common in such work, and arises from the presence in certain ores and fluxes of traces of that metal too minute for ordinary determination, but which make themselves felt in the aggregate.

tion of the air supply and, to prevent waste of heat on the one hand and to achieve a proper transmission of the heat of the ascending gases on the other, it should be kept at as high an elevation as is conformable with the annoying peculiarity of iron pyrites to agglomerate under the influence of a low heat, thus impeding the descent of the charge. Pyrrhotite ores and matte, when treated for concentration, are free from this drawback and can be run with a cold top.

"Pyritic ore furnaces usually exhibit the combustion of the sublimed sulphur a little distance above the stock line where the gases come into contact with the atmosphere; but this spurious 'over-fire' is not disconcerting, nor a real waste of heat. The latter arises when the gases and sulphur vapors, as they issue from the top of the charge, are too hot.

"The inner working of the pyrite furnace differs essentially from that of the ordinary coke-fed furnace smelting partially oxidized ores into matte. In the first place, there is, or should be, no oxidizing action anywhere except in the slag-forming zone itself. This is situated at a somewhat higher level than in the coke-fed furnace, especially when a cold blast is employed, and in it the oxidation of the special compound of iron and sulphur which arrives at this point, and the scorification of the iron, take place simultaneously. Below it there is no special chemical action and no further generation of heat, and above it there is none of the latter, but a decided progressive disintegrating alteration in the composition of the molten sulphide itself.

"The pyrite, after losing its volatile sulphur atom a few feet below the top of the charge, and being thereby transformed into a compound very similar to magnetic pyrites, is fused by the heat of the escaping neutral gases — which, in this region, are solely composed of nitrogen, sulphur dioxide and carbon dioxide (out of limestone and coke) — long prior to reaching any great depth. Subsequently, while coursing down the furnace to lower depths, it suffers a further desulphurization, without oxidation, simply under the influence of the gradually rising temperature. A roasting action does not take place anywhere.

"The particular combination of iron and sulphur which finally reaches the actual zone of simultaneous oxidation and slag-formation is generally lower in sulphur than the protosulphide. The whole action is self-directive or autonomous.

The silica remains chemically inert from the top down to the oxidation zone, and also below that zone, though exceedingly active within it. At the same time it is physically — that is to say, mechanically — of the utmost importance in these other levels through building up a honeycombed or vesicular aggregate, in the interstices of which the molten sulphide, blast-borne, trickles slowly down against the upcoming gases.

"In the 'laboratory' proper, roughly in the center of this porous structure, the mutual assimilation of the silicious pieces and the oxidizing sulphide corrosively destroy the individuality of both; but, on the one hand, there is a contraction of mass by the union of silica and protoxide for the formation of slag and, on the other, an explosive expansion of the gas volume due to the iron and the sulphur, assisted by the slag-formation.

"The condition of porosity is thus preserved in this particular spot by the action going on there, while it is the duty of the metallurgist to see that it is suitably maintained elsewhere in the furnace-shaft. The furnace must be kept properly open both above and below the fiery nucleus. In fact, it may be truly said that the proper maintenance of this purely mechanical feature is one of the principal secrets to success. In distinction from an opportune coke-bed for the charges to rest upon in their descent, there is only this silica skeleton to rely on for a proper furnace movement, and the situation is one far removed from a 'bed of roses.' Eternal vigilance is indeed required, and liberty from freeze-ups — and a minimum coke bill — can be secured for no other price.

"The special slag composition which falls is wholly subject to the self-governing relations which happen to be established between the air blown in and the iron sulphide and silica. The presence of coke and limestone, or of other substances yielding chemical bases, affects the result, but we can only consider the general features. The general tendency of the furnace conditions — as at Mount Lyell, for instance — is to form a singulosilicate, exactly as in the converter vessel. The introduction of heat from an extraneous source, such as through a heated blast, or by an addition of coke sufficient to live down to the oxidation zone, raises the acidity of the slag, but also decreases the calories developed by the sulphide, relieving and interfering with the work of oxidation. The matte then leaving the furnace is greater in quantity and lower in tenor than in the opposite case. The

matte formed is obviously only that part of the particular sulphide supporting the action in the oxidation nucleus which happens to be in excess of the influence of the blast. The copper, for its part, may be regarded as practically inert throughout the whole furnace action, and simply goes through the process under cover of the protection of the iron sulphide.

"Given a set adjustment of matter and force within the furnace, tending toward a certain slag-composition, the latter cannot be varied by decreasing or increasing the amount of free silica on the charge. It is true that this substance constitutes the direct available means for raising or lowering the degree of concentration of ore into matte; but an addition of silica is to be understood as having the effect of raising this degree without affecting the composition of the slag. There is *more* iron oxidized and slagged off, but the result is, not to make the slag more ferruginous, but simply to increase its quantity. Too great an excess of silica remains undigested, and chokes the furnace. Too small an amount simply causes a lower grade of matte to fall: *i.e.*, the amount of matte increases, which depresses its copper tenor, and the amount of slag decreases, but without altering its composition.

"This, in outline, is the action of the pyrite furnace. Needless to say, the practice varies in important respects in different localities. The chief point of difference, perhaps, is the use or non-use of a heated blast. A hot blast is only derivable from specially fired stoves, not from the furnace gases themselves, though a tepidly heated blast may be derived from hot-top furnaces. The extra paraphernalia and fuel-consumption introduced by the stove itself is, however, an economic encumbrance, so that after six years of experience it was gladly discarded at Mount Lyell in favor of a more energetic furnace manipulation, which was found to be capable of more than replacing the extra increment of heat supplied by the hot blast. This radical improvement was accomplished by simply supplying to four furnaces the entire quantity of air which had formerly been sent into eight. The effect was most marked in every direction. The rate of concentration and the furnace capacity increased, and costs decreased; also, essentially, the already small percentage of coke used. The present disposition is to regard the application of hot blast in the development of true pyrite smelting as a transi-

tional phase fully outgrown, and in the light of as unnecessary an adjunct as it is in converting.

"The grade of matte striven for naturally will depend upon its destiny and, if designed for converting, will vary between 35 per cent and 55 per cent in copper, depending on local considerations of cost and expediency. The original copper assay of the ore and the individual furnace practice decide whether the desired grade can, or is to, be achieved in one, two, or three smeltings. The concentration of low-grade matte presents no difficulties, and it early became an established rule at all works. Unfortunately, it appears to be so still, while, as a matter of fact, a single smelting should suffice, unless the ore is extremely poor. The single exception appears to be Mount Lyell. That establishment probably treats the lowest average grade of ore now being furnaced anywhere: namely, 2 to 2¼ per cent copper; but it concentrates this in the ratio of 18 or 20 to 1, with a single smelting and without difficulty.

"The whole question is one of a great blast supply, assisted by a low coke percentage, a high column, and constant watchfulness. Yet the blast delivery must not be excessive — merely adequate for the oxidation of sufficient of the sulphide. The Mount Lyell ore-supply permits of using about two parts of heavy pyrite to one of silicious material (containing 70 per cent silica), and it will be of interest to learn that, of the large amount of iron present, never less than 95 per cent is burned, or oxidized, the small remainder going into the matte. This oxidation factor is of great importance, since it controls the grade of the matte in copper. Under given conditions, the rise from $\frac{95}{40}$ per cent to 96 per cent of the iron enriches the matte from 40 per cent to 50 per cent of copper.

"With regard to the apparatus required for this process, it need only be remarked that the blast-furnaces used follow the general lines of construction characteristic of modern copper-matting furnaces, the most salient special peculiarity, perhaps, being a very large number of tuyeres. Other constructional demands which the furnace action would seem to make towards conformable departures from established types are not yet sufficiently clearly understood to have led to any positive innovations and the fixation of a special design, though, needless to say, there are innumerable inventions and tentative improvements before

the profession. The blast paraphernalia calls for no observation beyond the remark that its capacity should be ample, and that blast pressures are steadily rising, three pounds per square inch now being merely middling. As in ordinary copper smelting, rotary blowers chiefly continue to be used, though their place is disputed by piston-blowing engines, while the turbo-blower is the latest rival of both."

The fact that pyrite, when heated without air, becomes practically pyrrhotite — so far as chemical analysis can determine — would suggest that pyrrhotite itself would be an ideal sulphide for the pyrite process of smelting, and this inference is borne out by experience. The two companies working the great pyrrhotite masses near Ducktown, Tennessee, have experienced no more than ordinary metallurgical difficulties and, although preferring for the most part to use a two-stage smelting process in producing matte suitable for the converter, have found that it is quite practicable to accomplish this result in a single smelting, but not with entirely satisfactory slags as regards freedom from copper.

The concentration of low-grade matte in the pyrite furnace, with addition of silicious flux, is one of the simplest and most satisfactory instances of the application of the pneumatic processes to ores; for the low-grade matte, with its accompanying quartz, constitutes practically a charge of sulphide ore.

CHAPTER IX

PARTIAL PYRITE SMELTING

In the chapter on "Blast-Furnace Smelting" I pointed out that the advance in the art of metallurgy which has taken place during the past thirty years has extended this branch so widely that it no longer is convenient or logical to treat it as a single subject. To be sure, the eventual aim of the copper smelter, when fusing his ores in the blast-furnace, differs in no material degree from that of his most remote predecessor; both ancient and modern metallurgy had for its object the production of a more or less concentrated matte, or metal, suitable for further treatment, and of a worthless slag. The means by which this object is accomplished, however, have become so diverse that it would be incongruous and misleading to attempt to describe them without further classification.

We moderns have not discarded the ancient practice; we have merely added to it two new methods. For a considerable period of time we failed to appreciate the fact that we had actually introduced *two* new methods, and were trying to write about them as though there were only *one;* subject to modifications to be sure, but still based on the same general principles, and requiring merely a slight adaptation of the same treatment.

When I made the last thorough revision of old "Modern Copper Smelting," in 1894, I fell into this error — in good company — and practically divided the blast-furnace operations for smelting copper ores into two classes, as follows:

Coke Smelting	Pyrite Smelting
of oxidized (usually roasted) ores, including the melting-down of raw ores without any attempt to oxidize the constituents of the latter.	or fusing the raw ore by the heat derived (mainly) from the oxidation of its own constituents.

The first division of this classification was clear enough, but the second has caused great confusion in metallurgical literature

for years, and still exerts its obscuring effect upon many technical men who have not given especial attention to this subject.

In the present volume I am trying to make the matter more intelligible by refusing to treat of two diverse subjects under a single head, and I therefore eliminate "Pyrite Smelting" entirely as a title, and substitute for it the two headings, "*true* pyrite smelting"; and "*partial* pyrite smelting."

My present classification, then, will stand as follows:

Coke Smelting	True Pyrite Smelting	Partial Pyrite Smelting
of oxidized (usually roasted) ores, including melting-down of raw ores without attempt at oxidizing the latter during fusion.	the unroasted ore being smelted wholly (or partly) by its own heat, and so little coke being used that *it does not reach the oxidizing zone.*[1] Distinguished (almost invariably) by the fact that the metallurgist is trying to flux much iron with the smallest possible quantity of silica. Produces slags high in iron, and comparatively low in silica.	the unroasted ore being melted partly by its own heat, but with the addition of so much coke that *the oxidizing zone feels unmistakably the presence of carbonaceous fuel.*[1] Distinguished commonly by the fact that the metallurgist is trying to flux much silica with the smallest possible quantity of iron. Usually produces high lime-silica slags for commercial reasons.

The first of these methods is the ancient universal blast-furnace process for copper ores. It has been studied in Chapter VII. Its use is now greatly circumscribed, as we have learned that sulphides in our ores form too valuable a fuel in smelting for us to spend money in wasting them by roasting.

The second of these methods is invaluable in cases to which it is suited: namely, big bodies of almost massive sulphides in lump form. These are, however, of so rare occurrence that the entire world contains only a few plants of this nature. It has been studied in Chapter VIII.

The third of these methods is gaining ground with extraordinary rapidity, and already controls the situation in North America. The reason is obvious: almost every copper mine produces sulphide ores in lump form. When they contain too much gangue,

[1] See "True Pyrite Smelting," page 213.

we concentrate them and, reducing them to fines, remove them from the sphere of the blast-furnace; but when the sulphide lump ore is rich enough, we smelt it in the blast-furnace, and take advantage of its iron-sulphide content — be it much or little — feeling that, at the worst, we are escaping the roasting process and doing something toward lessening the coke bills, while — in favorable cases — we may reduce the consumption of carbonaceous fuel to a point that brings our process almost within the domain of true pyrite smelting.

It is truly extraordinary to see how the blast-furnace, which was once looked upon as the very type of everything that was *reducing*, is now familiarly regarded as the great *oxidizing* apparatus. This result has, of course, been brought about by increasing greatly the volume of air blown into the furnace within a given time. Whereas we formerly blew in just sufficient wind to burn the coke of the charge at the rate which was found most suitable for the existing conditions, we now blow in not only the air required to burn the coke, but also an additional quantity, so that there may be sufficient oxygen left to burn such proportion of sulphides as is necessary to produce a matte of the desired tenor in copper. Furthermore, as this combustion of sulphides (provided always that it occurs with sufficient rapidity, so that it shall be a bessemerizing instead of a roasting) evolves a large amount of heat, and produces a high temperature at just the point where it is desired, we are able to dispense with a certain proportion of coke, and thus gain the two great advantages which distinguish this method of smelting: abolition of the roasting process and plant; saving in carbonaceous fuel.

These great benefits, however, are obtained at the expense of two disadvantages: stronger blast, and more costly blowing apparatus; greater skill required.

The reason for the first of these disadvantages is obvious; it is, however, an insignificant factor in comparison with the saving effected in suitable cases.

The reason for the second disadvantage calls for a moment's consideration, as it is based upon the very keynote of the whole process. In the ordinary coke smelting of roasted sulphide ores, the materials that are fed into the blast-furnace are already prepared for the smelting process to a very considerable extent. They are not yet combined to form slag and matte; but the con-

stituents which are going to combine for this purpose are prepared so thoroughly that, if coke enough is burned to produce the necessary temperature, the desired result will certainly take place.

The most universal and important slag-forming reaction is the combination of silica with ferrous oxide. In the coke-burning furnace, the iron is already in the shape of an oxide, and requires but a trifling aid from the coke to become the oxide that is needed by the silica. Thus the composition of the slag is determined in advance, and is not subject to any vagaries of the process. The grade of the matte is also predetermined, and thus there are no unknown quantities to complicate the problem.

The matter is quite the reverse in the oxidizing smelting of unroasted sulphide ores. The white-hot free silica is present, looking anxiously for ferrous oxide in accordance with its nature; but the only iron present is a long way from ferrous oxide, being not only unoxidized, but in close union with sulphur. It is the chief aim of the partial pyrite process to unloose this combination of iron and sulphur by offering a superior inducement to the Fe in the shape of free oxygen, and at the same time to *fix* this new base — FeO — by providing it with a strong acid with which to combine the moment it is formed; hence the necessity of free silica. The residual FeS, which has not been provided with free oxygen and free silica, forms matte. Here, then, are two unknown quantities: ferrous oxide and matte. However, as the one is always the complement of the other, and as the proportion of ferrous oxide formed depends upon the amount of oxygen present, it is evident that the situation is controlled by the blast.

This difficult and sensitive variety of smelting, in which we create our iron flux out of the raw sulphide as we go along, is well termed "reaction smelting," and it is not difficult to see why it demands a higher degree of care and skill than in the older process where the more difficult and sensitive reactions were carried out in advance in the roasting process.

As I have already pointed out, the ores that are peculiarly likely to come to the partial pyrite furnace are lump ores from the mine, having a considerable proportion of silicious gangue, and tolerably rich in copper (perhaps five per cent at least), or gold and silver sulphide ores which will not concentrate without too much loss. Examining the preceding statement to understand the reasons for making it, we see that it must be lump ore — not

smaller than perhaps ½ inch (0.012 m.) diameter — in order not to choke the furnace, or make too much flue-dust; that it is likely to contain a considerable proportion of silicious gangue for the reason that — apart from great low-grade lenses — massive sulphides, in large quantities, are exceedingly rare; and that, when gangue is present with copper ore, it is highly silicious in the majority of cases; and that it is likely to carry a fair percentage of copper, else it would be classed as second-grade ore and sent to the concentrator.

Such a silicious sulphide lump ore as I have described is likely to be the chief material for which the blast-furnaces were built, and the question arises as to what we can use with it to neutralize the heavy excess of silica that it usually contains. This brings us to the almost universal flux (apart from iron) in partial pyrite smelting — limestone.

Although limestone will only yield about one-half of its weight in available CaO, its cheapness, general distribution, and ease of quarrying make it the almost universal flux in this variety of smelting. In certain districts where the ores are silicious, and the amount of sulphides small, the proportion of lime in the slag has been raised to an astounding figure, while the ferrous oxide has been reduced correspondingly. As already indicated, these high lime-silica slags are characteristic of the partial pyrite process, and demand especial attention from the metallurgist who expects to engage in this operation. They have been studied in some detail in the chapter on slags.

Magnesia, usually in the shape of magnesian limestone, may replace the lime to an extent that we should have thought impossible in former years. Twelve, and even fifteen, per cent of magnesia in a slag containing 18 per cent of lime seems to make no trouble, nor indeed to alter its character much from what it would be were lime the sole earthy base.

Barium sulphate, so unwelcome a substance in coke smelting, is thoroughly decomposed under the oxidizing influences and high temperature of the pyrite smelter, and makes a fairly good base for silica, although its high combining weight lessens its efficiency in lowering the silicate-degree.

While zinc blende is always troublesome, it is less so in the pyrite processes than in any other smelting method, being largely volatilized or slagged.

Arsenic and antimony, like zinc, are eliminated more satisfactorily by the pyrite processes than by roasting and coke smelting, their degree of removal, however, depending upon whether the particular case under consideration approaches the practice of true pyrite smelting, or whether a scarcity of sulphides forces it more toward coke smelting.

Having spoken of silicious lump sulphide ore, and of limestone — as the two most common and characteristic components of the partial pyrite charge — we may consider what other materials are likely to find a place in it. These materials are likely to be ores, as limestone is usually the only flux required, and they might be divided into two groups, according to whether their composition is such that they furnish more or less heat toward the smelting process, or whether they are inert in that respect, and simply have to be carried by the heat-producing constituents of the charge. These two groups, then, would be (*a*) ores containing appreciable amounts of metallic sulphides (arsenides, antimonides); (*b*) ores which do not contain metallic sulphides.

If the ore of the group designated (*a*) were in lump form and of a silicious nature, it would, of course, fall into the same class as the main silicious lump ore of the mine for which the furnace was built. If it were in lump form and of a basic nature, it would simply relieve us of using such limestone as would correspond to the excess base which this ore contained, while such sulphides as it might contain would be a welcome addition to our stock of fuel. This disposes of all ores of class (*a*) that are in lump form, and leaves for our consideration only those which are in the shape of fines — under which designation I will, temporarily, include all ores which would pass a ring having an opening of one-half inch (0.012 m.).

This matter of the use and disposition of fines is one of the most important practical subjects with which the blast-furnace smelter is called upon to deal, and probably occupies more of his time and attention than any other technical point connected with the smelting process.

The three principal sources of the fines which encumber every metallurgist who is trying to run a blast-furnace plant without the aid of reverberatories are: the fine material that results naturally from the mining, transportation, and breaking of the ore and of

which the lump smelting ore naturally contains its due proportion; concentrates from the mechanical concentration of the lower-grade ores; flue-dust, from the flues and dust chambers of the plant. It is not the place in this chapter to describe the methods that have been found the most suitable for consolidating these fine substances into a sufficiently stable form to withstand the somewhat strenuous treatment which they are forced to undergo before they have sunk deep enough in the furnace-shaft to be in safety. My aim at present is to point out what results are obtained in the blast-furnace in smelting this unpleasant class of material, whether as fines direct, or as prepared for the process by some form of consolidation.

Speaking first of the smelting of fines direct, without any consolidation or preparation, I will point out first a few of the disadvantages that they cause, and will then instance cases of the results obtained when running the furnace with an excessive proportion of fine ore.

The most obvious objection to this practice is the production of an excessive amount of flue-dust. Apart from the extensive arrangements required to catch all this fine material, and the loss and expense involved in handling it a number of times, the re-treatment cost soon becomes a serious item. It is manifestly ridiculous to charge it back into the furnace merely to see much of it blown out over and over again, and there comes a time when it is necessary to call a halt and effect something radical.

The question then at once arises whether it would not have been wiser to have effected this radical change at the very outset, and to have refused to feed the finest of the fines into the blast-furnace at all. This question has been settled, along general lines, by the united experience of metallurgists, and I think we may agree, without further argument, that with the heavy blast and rapid driving demanded by modern conditions, it is bad practice wilfully to feed into the blast-furnace materials below a certain size. I say *wilfully*, because there are many cases in which ores of suitable size contain a small admixture of unduly fine particles which it is not worth while to remove. It is better practice to put up with the disadvantage than it is to pass the entire mass of ore over grizzlies or screens for the sake of removing so small a proportion of fines.

We have now disposed of the lump ore and of the excessively

fine material, such as flue-dust, table concentrates, fine jig concentrates, etc., and have narrowed our field to granular material, the maximum size of which shall equal a circle of one-half inch (0.012) diameter, while its minimum allowable size remains yet to be determined. Hitherto, we have spoken of the disadvantages of fines merely in relation to the effect which they have upon the proportion of flue-dust that will result from their smelting. It is now necessary to examine their effect upon the smelting process itself.

Any attempt to discuss this subject with thoroughness would be out of the question in this book. Every practical smelter knows that if he has a charge consisting of moderate sized lumps of ore, without much medium-sized admixture, and adds a considerable quantity of fines, it will cause serious irregularities in the working of the furnace and lessen its capacity in a marked degree. The fines will sift down through the interstices between the lumps and, while the sulphide particles will do no harm, being speedily bessemerized, the particles of gangue (especially silica) will behave very much in the same manner in which sand used to behave in the copper converter when we injected it through the tuyeres to try to force it to combine with the ferrous oxide of the matte. The result was a sticky mass of slag, matte, and sand particles, and the upper portion of the focus of a furnace, suffering under the conditions which I have just described, presents an analogous appearance, as I know from repeated personal observation on the old brick Raschette furnace, where we could observe the interior at any time by pulling a few brick out of the breast.

This is the condition of affairs that results when fines are smelted in connection with lump ore which has only a small proportion of medium-sized fragments.[1]

The same addition of fines to lump ore that contains a fair proportion of medium-sized fragments works better, so far as the sifting down of the fines is concerned, the medium-sized pieces filling the interstices, and acting as the layer of ragging acts in a roast heap, where it prevents the covering of fines from sifting down through the main body of coarse fragments. But a new

[1] A typical illustration of this class of work is the smelting of converter-slag, in lump form, with raw sulphide fines, with the object of forming a low-grade matte to be added to the ore-charge in the regular blast-furnace work. In other words, simply a changing of the form of the sulphides into lump condition and, incidentally, cleansing the converter slag.

trouble now arises, owing to the fact that the smelting-column becomes too compact. The interstices between the larger lumps are pretty well closed by the medium fragments, and then are almost completely sealed by the excessive quantity of fines. The blast pressure goes up, often inducing the foreman to report that he is running with a powerful blast, while, in reality, the manometer pressure rises because there is no passage through the charge for the blast; the same result could be obtained by closing each tuyere gate until only one-fourth of the proper volume of air passed it.

This trouble was counteracted, to some extent, in our old brick furnaces by feeding the fines close to the walls, where the blast has always a strong tendency to seek the easiest escape, and thus force the air to seek its passage more toward the center of the ore-column. With modern water-jackets, however, this practice is usually impossible when using sulphide fines, as they have a tendency to agglomerate against the walls, forming annoying accretions which grow with rapidity.

It has long become evident that it is wiser to eliminate altogether an excess of fines than it is to struggle to counteract the evils that result from employing too large a proportion of this ever-present material. This matter has been studied most carefully at the Washoe smelter, and through the kindness of the company I can present certain results which are of permanent value.

The description of the regular blast-furnace work at Anaconda is given in another portion of this chapter; at present I am speaking only of the effect produced by fines upon the smelting process, and upon the production of flue-dust. A word as to the general conditions influencing the process at Anaconda is necessary for an appreciation of what is to follow.

The Anaconda blast-furnace mixture is extremely silicious, requiring an addition of over 50 per cent of limestone to produce a suitable slag. Coke costs 9 to 10 dollars per ton. It is evident that the two great demands of the process are iron-flux to replace the limestone in part, and sulphides to replace the expensive coke. Both of these demands are supplied by concentrated Butte sulphides, so that every endeavor is made to use the largest possible proportion of concentrates instead of sending them to the reverberatory smelters via the McDougall roasting furnaces.

All of the coarser concentrates are used, without preparation, down to those of three-eighths inch (0.009 m.) in diameter, the charge containing a sufficient proportion of comparatively large lumps (first-class ore and limestone) to keep the column open enough for rapid smelting. Moreover, a considerable proportion of these concentrates come from the "bull-jigs," which make a product up to 1½ in. (0.038 m.) in diameter.

Such of the fine concentrates as are used in the blast-furnace are contained in the briquettes that form so important a feature at this plant, constituting something like one-quarter of the entire blast-furnace mixture. Their composition varies from time to time, but the tendency is to increase the proportion of fine concentrates that they contain. This has been done cautiously for fear of delaying the smelting process and of producing too much flue-dust, but the importance of this addition may be seen by the fact that, during the first six months of 1909, the briquettes (dry) had the following composition:

screenings from first-class ore	23.29	per	cent
fine concentrates	33.40	"	"
rich slimes	3.01	"	"
pond slums	30.28	"	"
three-eighths inch concentrates	8.94	"	"
screenings from second-class ore	1.02	"	"
cleanings from old plant	0.06	"	"
	100.00	"	"

These briquettes had the following average analysis:

SiO_2	41.00
FeO	20.73
S	20.51
Al_2O_3	8.54
CaO	0.38
Cu	5.74
Ag (4.26 oz. per ton)	—
Au (0.29 oz. per ton)	—
	96.90

Later in the year, the composition of the briquettes was simplified, and the proportion of fine concentrates still further increased, the mixture in September, 1909, consisting of

first class screenings	25	per cent
pond slums	25	" "
fine concentrates	50	" "
	100	" "

This is an unusual proportion of raw sulphides to be held together by 25 per cent of concentrator slimes. It is rendered possible by the remarkably adhesive properties of the slimes from the concentration of the Butte ores, resulting from the kaolin and other products of the decomposition of feldspar, that they contain.

These briquettes contain from 12 to 18 per cent water, and are fed moist into the blast-furnace. Their influence upon the production of flue-dust is an important question.

The results at Anaconda show that about six per cent of the weight of this latter class of briquettes passes off in the shape of flue-dust, this being 12 per cent of the fine concentrates that they contain. The retreatment of this material, however, is appreciably cheapened by the fact that about 83 per cent of this flue-dust consists of thoroughly oxidized fine concentrates which go to the reverberatory furnaces as a welcome material, being preferred even to the roasted concentrates from the McDougall calciners. About 1000 tons daily of briquettes is used at the blast-furnaces, and the flue-dust resulting therefrom averages a weight of 60 tons. Apart from a marked saving in coke, the influence of this increase in the use of fine concentrates shows plainly in the increase of FeO in the blast-furnace slag. This has risen to 22 per cent from a former 17 to 18 per cent.

The influence of fine pyritous concentrates upon the production of flue-dust in the blast-furnace is so important a matter that exact records, covering a period of over six years and representing several million tons of total furnace-charge, cannot fail to be interesting. The following table from the records of the Washoe smelter shows this point clearly, and shows also that, when the pulverized sulphides are well briquetted, their influence upon the production of flue-dust is not nearly so great as might naturally be expected. This table is based upon the total charge of the blast-furnaces, including the limestone.

Year	p. c. Briquettes in Charge	p. c. Fine Concentrates in Briquettes	p. c. Flue-Dust
1903	6.48	—	2.29
1904	11.66	—	2.21
1905	18.32	14.19	2.72
1906	21.10	27.77	2.69
1907	26.38	36.57	2.73
1908	25.02	11.97	1.62
1909 6 months ..	23.40	33.40	2.56
1909 April	27.16	45.58	3.00
1909 May	25.66	42.51	2.71
1909 June	27.57	38.38	3.59
1909 July	26.55	43.56	3.32

Through the courtesy of H. A. Keller, superintendent, I received in 1895 a statement of the average results of the blast-furnace work at the Parrot smelter in Butte for the year 1894. While the advance in the art of metallurgy during the intervening years has been so great as to render most of the features of this statement valueless to the metallurgist of the present day, it presents one point of permanent value which warrants its inclusion in the present work. This is the record of the proportion of flue-dust produced when running on a charge consisting of 60 to 70 per cent of fines, most of them having undergone a previous roasting.

One other point is also interesting: namely, the distinctly oxidizing nature of the fusion, by which 66 per cent of the sulphur in the charge was removed during the melting in the blast-furnace, and a 50 per cent copper matte was produced from ores carrying 14.6 per cent sulphur, with 10.64 per cent copper. The proportion of coke used also was low, showing that a considerable benefit was derived systematically from the oxidation of the comparatively small proportion of sulphide ores that had evaded the roasting processes.

Although imperfect and wasteful, this is a distinct case of partial pyrite smelting, the oxidizing effect being induced mainly by the extremely low ore-column and hot top. The result is a roasting of the sulphides in the upper portion of the shaft rather than a bessemerizing of them at the focus, which is more the condition that we desire to approach in modern practice. It will be remembered, of course, that these were the small water-jacketed

furnaces of the earlier type, taking only about one ton at a charge. The comments upon this statement were written by myself in 1895, and reflect the opinions of that time.

BLAST-FURNACE CHARGE, 1894

DEBIT	gross lb.	H_2O %	net lb.	Insol. Residue %	Insol. Residue lb.	FeO %	FeO lb.	CaO %	CaO lb.	S %	S lb.	Cu %	Cu lb.
1. Calcines	694	21.0	548	16.7	91.52	47.6	260.85	—	—	9.0	49.32	13.7	75.08
2. Concentrates,etc.	106	3.0	103	21.5	22.14	38.8	39.96	—	—	0.1	38.21	9.1	9.37
3. Kiln ores	398	11.0	354	34.0	120.36	39.9	141.25	—	—	25.5	23.01	9.0	31.86
4. Custom ores	366	3.0	355	46.0	163.30	19.3	68.51	—	—	.0	88.75	8.0	28.40
All ore	1,564	—	1,360	—	397.32	—	510.57	—	—	—	199.29	—	144.71
5. Converter slag	436	—	436	40.3	175.71	51.9	226.28	—	—	—	—	1.4	6.10
6. Scrap copper	10	—	10	—	—	—	—	—	—	—	—	90.0	9.00
Ore and slag	2,010	—	1,806	—	573.03	—	736.85	—	—	—	199.29	—	159.81
		CO_2											
7. Limestone	100	40%	60	2.0	2.00	—	—	52.0	52.00	—	—	—	—
8. Montana coke	198	80%	40	20.0	39.60	—	—	—	—	—	—	—	—
Total charge	2,308	—	1,906	—	614.63	—	736.85	—	52.00	—	199.29	—	159.81
Less "credits"			492	—	51.84	—	175.70	—	2.55	—	69.44	—	151.43
			1,414	—	562.79	—	561.15	—	49.45	—	129.85	—	8.38
Slag actually made			1,210	46.5	562.65	46.4	561.44	4.1	49.61	—	—	0.3	3.62
Unaccounted for			204	—	0.13	—	0.29	—	0.16	—	129.85	—	4.76
CREDIT													
1. Matte			237	1.0	2.37	28.3	67.07	—	—	22.2	52.61	49.8	118.03
2. Flue-dust			255	19.4	49.47	42.6	108.63	1.0	2.55	6.6	16.83	13.1	33.40
Total			492	—	51.84	—	175.70	—	2.55	—	69.44	—	151.43

Slag

SiO_2	36.2
Al_2O_3	10.3
FeO	46.4
CaO	4.1
	97.0

LOSSES IN SMELTING

Net	= 204	pounds	= 10.6	per	cent	of	net	charge
S	= 129.85	"	= 65.9	"	"	"	"	"
Cu	= 8.38	"	= 5.2	"	"	"	"	"

"The most noticeable feature in this table is the enormous production of flue-dust, amounting to 18.75 per cent of the ore charged.

This arises mainly from the great quantity of fine ore used, owing to the charge consisting of calcines, which are both fine and light; of concentrates and screenings, which are mostly fine; of kiln, or rather, stall-roasted ore, which is exceedingly pulverulent; and of custom ores, which contain a variable, but very large, proportion of fines. Some 60 to 70 per cent of the ore charged into the cupola is thus in a pulverized condition. This necessitates running with a low ore-column, a circumstance which not only produces much flue-dust, but also keeps the tunnel-head very hot. The result of this combination of circumstances is slow smelting, low ore-column, and considerable volume of wind; and this produces the oxidizing effects so often and strongly insisted upon throughout this work." (Old "Modern Copper Smelting.") As may be expected, the oxidation is excessive, no less than 66 per cent of the sulphur being removed during this fusion. It will be noticed that all of the ores treated are comparatively high in sulphur for blast-furnace work; thus it becomes essential to remove much of the sulphur in this operation.

This study of the influence of fines upon the production of flue-dust has not yet led to any arbitrary statement as to what may be the smallest sized material that it is judicious to feed into the blast-furnace, or what is the largest size that it is worth while to briquette, agglomerate, or otherwise compact. As a matter of fact, no such statement can be made owing to the diverse influences which modify each separate case.

Considering a large number of cases, representative of modern methods, I am inclined to think that we need not fear any serious clogging effect of material of one-half inch (0.012 m.) diameter, unless the rest of the charge also consists of fragments only a little above this in size. When the proportion is not too great, I should not hesitate to charge material of one-fourth inch (0.006 m.) diameter. Below this size, it seems wiser to compact the fines by one of the various devices described in another chapter. This is necessary not only on account of the rapid diminution in the rapidity of the smelting process when clogged with fine material, but also to avoid the production of an excessive amount of flue-dust. There will always be a considerable proportion of fines, running down in size to the finest dust, even in the lump ore, and it is usually bad policy to add any more. It is cheaper to compact

them, and thus lessen the production of flue-dust, and make the blast-furnace work much more rapid and comfortable.[1]

The subject of Partial Pyrite Smelting covers a broad field, lying between two boundary lines which cannot be accurately established by mere casual observation.

It treats of the smelting of raw (or even partly roasted) sulphide ores in the blast-furnace processes where more air is provided than is necessary to burn the coke, and, if we stopped at this point, the definition would be comparatively simple, as this lower boundary line is not difficult to recognize. But such a definition would also include *true* pyrite smelting, and would thus bring under one head methods which are too different in their nature and results to bear such intimate association without confusion.

I have, therefore, undertaken to simplify the examination of these two distinct operations — true pyrite smelting and partial pyrite smelting — by drawing a line between them at a point established by natural laws and consequently easy to determine on paper, but very difficult to recognize in practice without the aid of more elaborate research than is compatible with the overdriven work of the ordinary smelting plant.[1]

This division between true pyrite smelting and partial pyrite smelting comes, as I have already indicated, at that point where so much coke is used in the charge that a portion of it is unconsumed by the SO_2 gas (which is the sole oxidizing agent in the upper portion of the true pyrite furnace-shaft),[2] and descends far enough into the true oxidizing-zone (focus) of the furnace to make its influence distinctly felt by diminishing the supply of oxygen available for the combustion of the sulphides, and thus lowering the grade of the matte, as well as diminishing the proportion of FeO in the slag.

Owing to what may be termed the normal irregularities of furnace practice, it is impracticable to establish this division with exactitude, and we may occasionally encounter an instance where we must admit that we are on debatable ground and cannot determine to which division the process under examination belongs. Indeed, it is quite conceivable that the unavoidable fluctuations of the furnace work may swing it backward and forward across the boundary line, so that in the morning it may be a true pyrite

[1] See Chapter XIII for methods for compacting fines.

[2] See chapter on "True Pyrite Smelting."

smelting operation, while in the afternoon a lower-grade matte and more silicious slag may indicate its temporary admission to the partial pyrite class

The mere question of nomenclature, however, is of little moment, provided we appreciate the true reasons for our uncertainty. Intelligent doubt is infinitely removed from unreasoning certainty. Moreover, it happens — from causes entirely alien to the province of metallurgy — that there are very few cases in practice which approach at all closely to this division line. True pyrite smelting demands great quantities of almost massive sulphides carrying sufficient metal values to pay for treating, and there are only a few such deposits in the world, as yet known. Omitting these few exceptional cases, we drop at once to sulphide ores which are far from massive or, even if massive, are swamped in the bulk of other more valuable silicious ores to which the iron of the solid sulphides is a much needed flux. Consequently, in these cases we rob the comparatively scanty massive pyrite ores of their ability to fuse themselves, by adding to them a large amount of silicious sulphide (or non-sulphide) ores, which need the iron of the massive ores, thus producing a mixture far too low in combustible substances to smelt without the addition of coke to such an extent as to carry it well over the division line between the two methods. This dilution of the sulphides is generally still further increased by the addition of the considerable proportion of limestone that is necessary to neutralize the abundant silica of the added ores.

Even with the contraction of the field of study to the area just described, we find that our "partial pyrite smelting" has still too wide a scope, and includes too many utterly diverse situations and conditions to enable us to base definite conclusions or to formulate exact laws from its examination. We now embrace, under this title, almost all important smelting of copper sulphide ores that is executed in North America, excepting Sudbury (the representative of the old method of roasting and coke smelting), and Ducktown and Kennett (the representatives of true pyrite smelting).

This covers such a number of widely diverse conditions, and such variety in practice and results, that the image derived from their contemplation is too confused and unsuitable for clear deductions.

Most metallurgical writers would gladly avail themselves of any rational classification by which this mass of heterogeneous material might be simplified and made more available for study, while, at the same time, opportunity was given for the deduction of general principles which might be of great practical value to the operators themselves. This desirable simplification seems to me, however, to be impossible at the present time. We cannot generalize, and classify, and deduce, without the basis of a considerable groundwork of facts, and we lack almost absolutely the most essential facts concerning this subject.

The especial facts which we lack so completely, and which we need in every case before we can form a true estimate of the reactions taking place within the furnace, are two in number: (*a*) average analysis of the blast-furnace gases at different horizons in the shaft, sufficient in number, and covering sufficient periods of time, to give us a true conception of their composition.[1]

The remaining factor that we usually lack in studying the reactions within the blast-furnace is (*b*) sufficient determinations, in each case, to ensure an exact knowledge of the volume of air entering the furnace through the tuyeres. At most plants admirable records are kept as to the number of blowers at work, displacements, revolutions per minute, blast pressure at each furnace, composition and analysis of charge and products, etc.

All these are admirable so far as they go, but without a knowledge of the exact composition of the furnace gases, they are, for scientific purposes, much like some ancient historical inscription in a lost language — full of possible information if we only had the key; and the key, in our case, is the composition of the furnace gases at various horizons.

Even the blast records are more often misleading than helpful.

[1] When it is intended that the gas analyses shall represent the working of the regular process, the greatest care must be taken to obtain the samples of gases at times when the furnace is running in perfectly normal condition. Even slight irregularities are likely to cause the most astonishing and contradictory variations in the composition of the gases, and any attempt to average up these irregular results with the normal ones will be like adding dollars to francs, and expecting to draw deductions from the result.

This withdrawal of representative gas samples at suitable times demands unusual care, ingenuity, and patience, and is a branch of research that can be recommended strongly to our great metallurgical plants as a subject that will benefit both science and process.

Without speaking disrespectfully of any type of blowing apparatus, I will state from personal experience, fortified by ample testimony from unexceptionable sources, that the quantity of air actually entering the furnace — even in well-conducted plants — bears so little relation to the speed and displacement of the blowing apparatus that any estimate of air based upon this factor would be extremely misleading. This is especially the case where rotary-piston blowers are used for comparatively high pressures, or where reciprocating piston blowers are used for very low pressures. I speak of this subject not to criticize our present blowing apparatus, which seems reasonably good for its first cost, but to point out the pity of invalidating costly and laborious investigations by basing them upon an uncertain and fluctuating foundation.

The actual measurement of the air-current is the only way to establish a factor of correction, and even this determination must be repeated occasionally to allow for the decreasing efficiency of the blowers as they become worn.

It follows, therefore, that instead of discussing partial pyrite smelting as a definite operation from which we can deduce certain general principles and formulate certain general rules — as we can do, to a considerable extent, in true pyrite smelting — we must content ourselves for the present with classifying such facts as we possess, and pointing out the conditions that seem favorable for the production of certain results, without understanding fully the means employed by the furnace to effect these results.

As an illustration of my meaning, I will ask a few questions, to which I am unable to return answers based upon positive knowledge, and consequently am unable to offer any satisfactory description of the chemistry of the process.

Question 1. — Upon what basis is the oxygen of the blast divided between the coke and the melted sulphides in the focus of the partial pyrite furnace?

There is no use in replying that the coke has the first call on the oxygen and the sulphides take what there is left, else why should there be abundant oxygen still left in the upper regions of the shaft (as any one may see) while there is still an excess of sulphides remaining to form matte? In the true pyrite furnace, where there is not even any coke in the focus to steal oxygen, the molten sulphides take it up so thoroughly that numerous analyses have

practically failed to show its existence above the focus in normal furnace work.

Question 2. — What proportion of the sulphides simply undergoes roasting in the upper part of the shaft? What proportion is bessemerized in the more active zone? What proportion escapes both of these influences? What determines these proportions? [1]

Question 3. — Is SO_2 neutral or oxidizing to coke in the upper regions of the normally-running partial pyrite furnace?

This is an illustration of a large number of such questions that we cannot answer for want of the knowledge to which I have just referred, and without which it is a waste of time to argue about the chemistry of partial pyrite smelting.

This discussion of our ignorance in the matter must, in part, take the place of a more scientific description of the process, and a study of a few of the representative modern plants will bring out the extensive information that has already been accumulated,

[1] It is, of course, an undeniable fact that a considerable amount of roasting of the sulphide occurs in the partial pyrite process. This is shown by the lowering in the grade of the matte which occurs when lump sulphides are substituted for finer sulphide material. This roasting dissipates, in the upper portion of the shaft, heat which could be better applied at the focus, besides causing a hot top, with its attendant train of evils.

The question is: how far is this roasting essential to the process? Why cannot we establish a concentrated focus, as in the true pyrite furnace, and burn all of our oxygen within these limits, expecting a certain proportion of the coke to be oxidized in the upper regions by reducing SO_2 to S, while the remainder of it enters the focus and aids the scanty pyrite in maintaining a suitable temperature?

The coke in the focus will have the prior claim on the oxygen of the blast, and there should be just enough oxygen remaining to bessemerize such proportion of the sulphides as is required to produce the proper grade of matte.

Then our partial pyrite smelting would approach in its behavior to true pyrite smelting, modified only by the fact that both coke and sulphides will be burned in the focus, instead of sulphides alone, and we shall have a process much less wasteful of heat than the present exceedingly mixed combination of coke-burning, roasting, and bessemerizing.

I do not suggest that such an ideal situation is possible, but it seems to me that we can make no great advance in the economy of the present process, until we learn to surmount the difficulties which arise from carrying an ore-column of 15 to 20 ft. in height, with corresponding power behind the blast.

and will serve to illustrate the work about the furnace, as well as the construction of such details of the plant as have not yet been noticed.

Beginning with the plant whose enormous capacity, exact records, and liberal attitude toward science make it the natural object of study for writers on the metallurgy of copper, I will point out the more interesting details of the blast-furnace work at Anaconda, and will mention such points in furnace construction or management as seem particularly useful.

The ore which comes to the Washoe smelter at Anaconda is produced, for the most part, by the company's own mines at Butte, the distance from the mines to the smelter-bins being 30 miles. The cost of delivery over this distance is 12 cents per ton.[1]

The accompanying flow-sheet shows the progress of the various classes of ores through the Washoe smelter.[2]

The blast-furnaces handle the following material: first-class ore, briquettes, concentrates, matte, rich slag, limestone, coke; at times, small amounts of copper precipitate are added, from the Butte mine waters.

First-class ore, at the Washoe smelter, means ore unsuitable for concentration, and is usually — though not necessarily — richer in copper than the second class, or concentrating, ore. Comparatively rich silver ore, high in blende, and often low in copper, is added to this class. For the most part the cars, containing this ore, dump it over grizzlies so that only the lump ore goes direct to the blast-furnace bins, the fines being added to the briquetting mixture. The first-class ore, as a rule, contains a large amount of silicious gangue, and usually constitutes about 20 per cent of the total furnace-charge.

The briquettes have just been described at length. They vary in composition, but, with the present practice, are intended to contain as large a proportion of fine pyritous concentrates as their structure will permit. At the time of this description they consist of one-half fine concentrates, and one-fourth each of first-class screenings and concentrator slimes from the settling ponds.

[1] One ton = 907.2 kg. One foot (12 inches) = 0.305 m. One oz. per ton (Au or Ag) = 0.0034286 per cent.

[2] This flow-sheet is taken from a description of the Washoe smelter, by one of the staff, in "Mines and Minerals," October, 1907.

They constitute something over 20 per cent of the total furnace-charge.

The concentrates are the granular pyritous material resulting from the jigging of the second-class ore, and run down to $\frac{3}{8}$ inch (0.009 m.) in size. They usually form about 15 per cent of the charge.

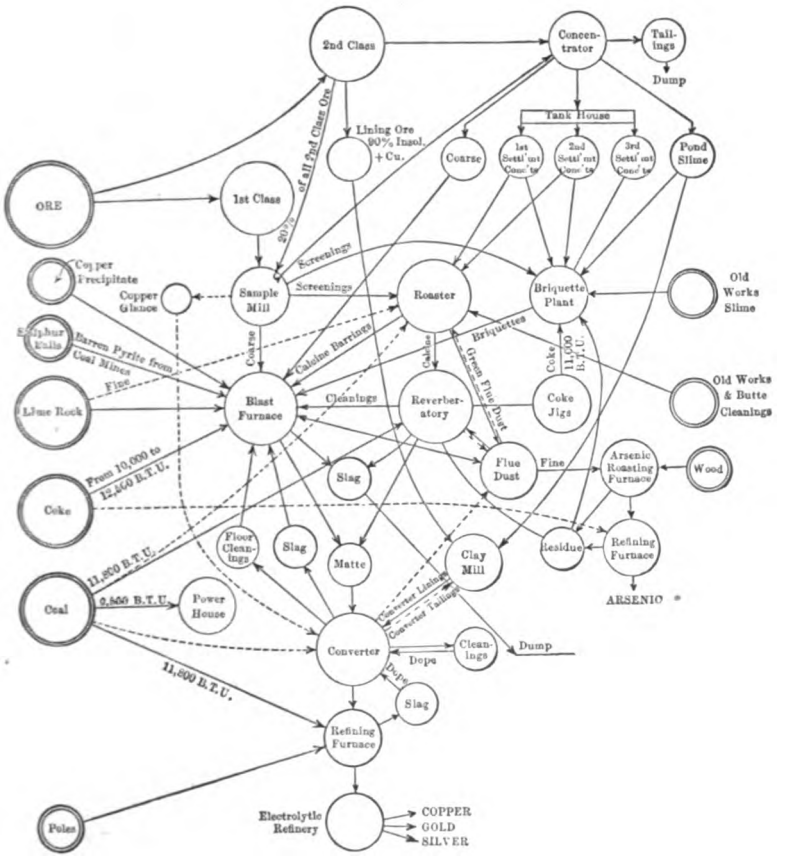


FIG. 52. — Washoe flow-sheet

The matte and rich slag are simply products of this nature, from various operations, which are returned to the blast-furnace for re-working. The converter-slag — averaging 2½ per cent copper — is the most important of these as regards quantity. Spilled matte, ladle skulls, runners, and similar rich material are

included under this head. The slag and matte together may constitute 13 or 14 per cent of the total charge.

The limestone forms an important addition to the charge, as the ore-mixture proper is very silicious. In order to bring the slag down to about 40 per cent silica, it is necessary to use something like 32 per cent of limestone. This is one of the most striking features of the Washoe practice. With an exact knowledge of the advantages and disadvantages that will result from cutting off a single hundred-weight of this limestone, they find it more profitable to employ this excessive amount of barren flux than to attempt to run with a slag higher in silica.

The limestone contains

SiO_2	5.76
FeO	0.57
Al_2O_3	1.14
CaO	49.13

Some 2000 tons daily of this limestone is used in the blast-furnaces at a cost of about 50 cents per ton.

Various makes of coke are used, the average price being about $8.75 per ton. The following analysis gives a general idea of the composition of the coke:

moisture	1.91	per	cent
volatile matter	1.39	"	"
fixed carbon	80.24	"	"
ash	16.46	"	"
	100.00	"	"

The proportion of coke used varies with the composition of the charge and the condition of the furnace. An increase in the sulphides permits a decrease in the coke. At present writing 8.2 per cent of coke is employed. That is to say: a charge of 12,200 lb. of ore, briquettes, matte, slag, concentrates, and limestone is receiving 1000 lb. of coke of the above composition. This indicates, of course, that a considerable proportion of the heat needed for fusion is being derived from the oxidation of the sulphides.

The following analysis indicates the average chemical composition of the entire blast-furnace charges (excepting the coke) for a period of several months:

Cu	5.21	per cent
SiO_2	26.41	" "
FeO	18.60	" "
S	11.46	" "
Al_2O_3	4.26	" "
CaO	17.49	" "
	83.43	" "

The CO_2 belonging to the limestone will account mainly for the balance.

This charge contained 3.65 oz. silver and 0.023 oz. gold per ton.

The following average analysis shows the proportion of the more important constituents contained in the resulting matte:

Cu	46.34	per cent
Fe	24.50	" "
S	24.00	" "
SiO_2	0.30	" "
	95.14	" "

This matte contained 33.28 oz. silver and 0.201 oz. Au per ton.

The average composition of the blast-furnace slag for a long period of time was:

SiO_2	39.87	per cent
FeO	22.38	" "
CaO	26.83	" "
Al_2O_3	6.35	" "
ZnO	1.40	" "
S	0.52	" "
MnO	0.60	" "
Cu	0.30	" "
	98.25	" "

This slag contained 0.12 oz. silver and 0.0006 oz. gold per ton.

The composition of the furnace-charge must, of course, vary according to fluctuations in the delivery of ores and concentrates. At this smelter a "charge" means the amount of mixture that is fed into two furnace-doors. The furnaces are fed on either side alternately, and the smaller furnaces have six doors on either side; the larger furnace, ten doors. The furnace-charge (for two doors) in use at the time of writing this description consists of

coke	1000 lb.
coarse concentrates (⅜ in. and upward)	1800 lb.
first-class lump ore	2400 "
slag and matte	1600 "
briquettes (wet weight)	2500 "
limestone	3900 "
	12,200 "

The average sulphur content of this charge is 11.5 per cent, and 78 per cent of this sulphur is removed during the fusion. The average per cent of copper in the matte is 46.6 per cent.

A negative pressure of 0.7 inch in the flue removes the fumes thoroughly.

The low iron and high lime content of the slag is especially noticeable. In the course of experimenting, the FeO in the slag has been run down as low as 17 per cent, with 30 per cent CaO and 41.5 per cent SiO_2. This made a very clean slag running sometimes below 0.25 per cent copper, but was inclined to chill, both at the tuyeres and in the settlers.

During these same experiments, it was interesting to observe the interchangeability of coke and sulphides within a tolerably wide range. The following statement refers to the percentage of sulphur which, of course, corresponds to a much greater weight of the sulphides: so long as the percentage of sulphur in the charge plus the percentage of coke equals 18, it makes no perceptible difference in the capacity and general behavior of the furnace whether the coke was as high as 11 or as low as 7. In other words, 7 coke + 11 S (as sulphides) = 11 coke + 7 sulphur. As these experiments were conducted by experts, and covered large amounts of material, they give us a valuable basis for forming some idea of the relative calorific value of sulphides in practical work. The results, however, are modified by the fact that, as the proportion of sulphides is increased, the proportion of FeO in the slag increases correspondingly. This increases the fusibility and general good qualities of the slag to a considerable degree, and thus destroys our basis of comparison.

The former blast-furnace plant of the Washoe smelter consisted of seven furnaces in a row, situated with their long axes parallel to the length of the building, and having an inside section, at the tuyeres, of 56 inches by 15 feet. As increased capacity was required, Mathewson threw two of these furnaces into one

by removing the vis-à-vis end-jackets, and enclosing the space that was included between the now open ends of these two furnaces. Thus, from two 15-ft. furnaces and their intervening space, he formed one large furnace having the same (56 in.) width as the old ones, but possessing an inside length of 51 feet. In this manner, two new large furnaces were formed from four small ones.

Finding that the large furnaces were easier to manage and more profitable to run than the small ones, and satisfying himself that there was no technical limitation to the length of such a furnace, Mathewson merged the three remaining 15-ft. furnaces into one furnace having an inside measurement of 87 feet. As further capacity is demanded, this large furnace is now being extended an additional 40 feet. To distinguish between them, the 87-ft. furnace is known as the "large furnace," while the 51-ft. furnaces are denominated the "small furnaces."[1]

It is easy to understand the manner in which the melted products are discharged from these long furnaces by noting that the trapped discharge-spouts for slag and matte are situated just as they were formerly when the plant consisted of independent small furnaces. Thus, a 51-ft. furnace — which is made up of a small furnace on either side and the intervening space between, has two discharge-spouts, one each for the two original small furnaces, while the central section has no discharge-spout, but is provided with a somewhat higher bottom, which slopes either way toward the end sections. The 87-ft. furnace, made up of three small furnaces and two intervening *spaces*, has, of course, three discharge-spouts — one to each of the original furnaces — while the more elevated bottoms of the filled-in sections slope either way.

The 51-ft. furnaces have 88 four-inch tuyeres, the 87-ft. furnace having one hundred and fifty.

It need scarcely be said that the management would never have adopted a method of construction which concentrates so

[1] The accompanying cuts, taken from the patent specifications, give a good idea of one of the 51-ft. furnaces.

A clear and correct description of the Anaconda furnaces, and their method of operation, is given by C. Offerhaus in the *Engineering and Mining Journal* of Aug. 7, 1909.

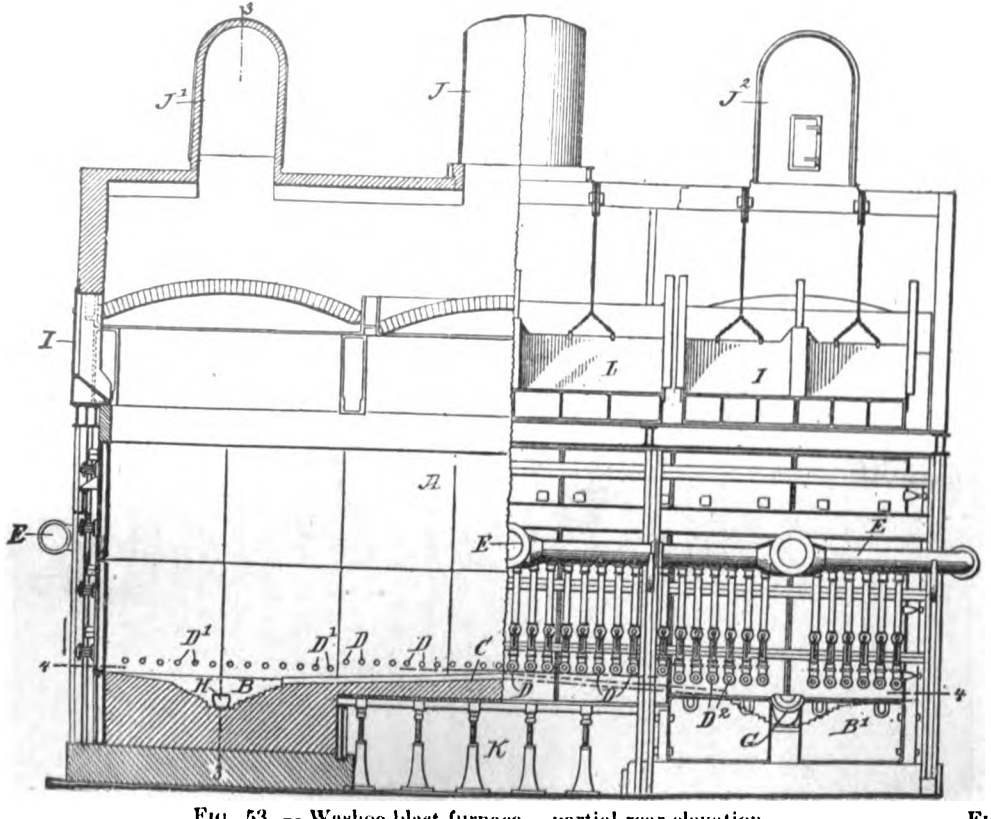


FIG. 53. — Washoe blast-furnace — partial rear elevation

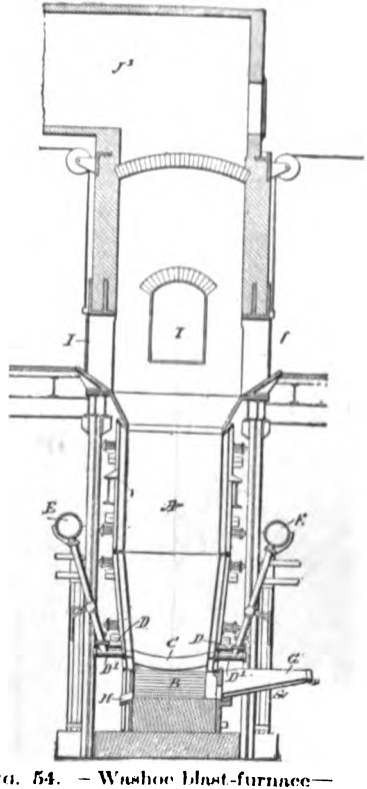


FIG. 54. — Washoe blast-furnace—

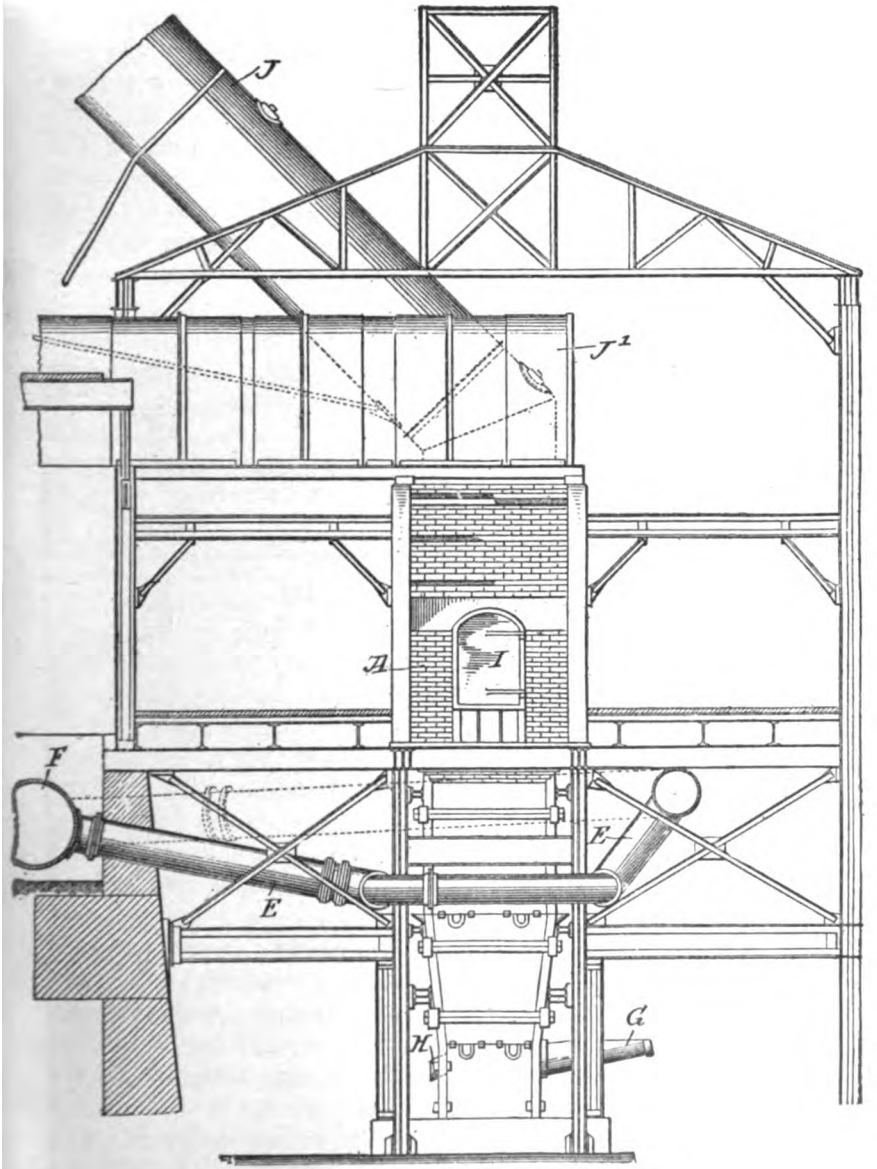


FIG. 55. — Washoe blast-furnace — end elevation

large a proportion of total capacity into single units, if it had not first satisfied itself fully of the feasibility of executing extensive repairs without shutting down more than the section of the furnace directly involved; thus these furnaces do not blow out for repairs.

A leaky jacket, for instance, is replaced by shutting off the tuyeres in the faulty jacket as well as in the one directly opposite, while the cooling water is allowed to circulate freely until the buckstays, pipes, etc., have been taken down, and the jacket is ready for removal. This chills a wall inside the jacket strong enough to close the opening safely. The new jacket being placed in position and pipe connections made, the temporarily idle portion of the furnace soon resumes its functions. The entire end

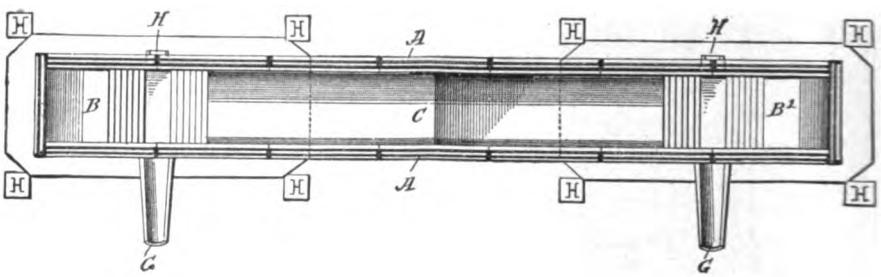


FIG. 56. — Washoe blast-furnace — sectional plan on line 4–4 of Fig. 53

of a furnace has been shut down, cleaned out, and undergone extensive repairs — occupying some 10 days — while the remainder of the furnace was in full operation.

Apart from the saving in space, some of the chief advantages of this large type of furnace are: increased hearth area without increasing the area of the ends, thus having a proportionately smaller radiating surface; comparative freedom from crusts and accretions, which form and cling most obstinately at the right angles formed by the sides and ends of the shaft; saving in coke, amounting to between one and one and one-half per cent of the weight of the ore-charge; saving in labor; increased capacity per sq. ft. hearth area; remarkable flexibility, evinced by the ease with which a section is shut down for repairs without interfering with the remainder of the furnace.

While sections corresponding to the former small furnaces are built up solid from the ground, the new filled-in sections have a

bottom of silica brick laid on water-cooled cast-iron plates mounted on cast-iron columns.

The 51-ft. furnace is provided with three seven-foot unlined steel flues, which go off at an angle of 45 degrees, and discharge into a large brick and steel dust chamber which connects with the main flue. The 87-ft. furnace has five of these steel flues.

The larger furnace smelts from 2500 to 3000 tons mixture per 24 hours, the two 51-ft. furnaces smelting about 1600 tons each. This makes some 6000 tons daily of material to load, weigh, convey, and feed into the furnaces, being an average of 250 tons per hour. The manner in which so large an amount of diverse materials is handled in so confined a space is worth examining, as it is by such organization, and by the sweeping substitution of mechanical appliances for hand labor, that modern practice has become possible. The mere enlargement of furnaces, remarkable as it may seem, is perhaps the least radical portion of this development.[1]

The blast-furnaces are charged, alternately on either side, by a train of eighteen 3-ton cars, running on tracks close alongside the charging-doors. This train is handled by a compressed-air locomotive, operating under a pressure of 1000 lb. per sq. in., and is loaded, by gravity, from the storage bins containing the various materials which go to make up the mixture. These bins are in three rows, each row being 28 ft. wide, 786 ft. long, and 20 ft. deep, and act as storage for first-class lump ore, coarse concentrates, lime rock, converter-slag, etc. They are filled from the full-sized steam trains which run above them, and their gates are operated by compressed air. The cars, while being filled from the bins, rest on track scales; and the materials of the charge are run into the cars in the following order: converter-slag, lump ore, coarse concentrates, limestone. The train then moves under the hoppers of the briquetting plant, where, with a single movement of a lever, each hopper drops its burden of moist briquettes into the car beneath, the 18 cars thus being loaded simultaneously.

[1] In connection with personal notes, I have made use of a concise official descriptive pamphlet, compiled by members of the Anaconda company's staff, and entitled, "A Brief Description of the Washoe Smelter," Anaconda, Montana, 1907. The following cuts are from the *Engineering and Mining Journal.*

The coke is charged in large steel barrows, containing 1000 pounds, this constituting a charge for two furnace-doors, although the proportion charged into each door varies with the condition of that particular section of the furnace. The ore train being placed in position at the furnace, the heavy iron doors are lifted by compressed air, and the side-dumping cars discharge their loads, being tipped by means of compressed air, at 90 lb. per square inch. A train of 18 cars is discharged every 10 to 15 minutes.

There is nothing particularly noticeable in the management of these large furnaces. The great volume of hot material tends to

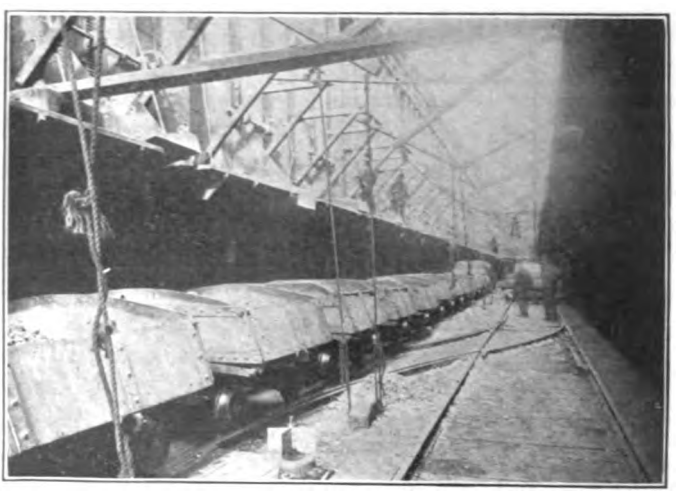


FIG. 57. — Washoe smelter charging train receiving load from ore-bins

equalize irregularities and to minimize the influence of local disturbances. The tuyeres are punched regularly every four hours with a bar of $\frac{3}{4}$ in. (0.019 m.) diameter, those which cannot be opened being removed and drilled clear with an air-drill. The end-tuyeres are seldom used, as they tend to cause local chilling.

The continuous discharge of the slag and matte from the furnace takes place through a water-cooled trap-spout, which is bolted to the breast-jacket, and also supported from below. With the high-grade matte produced at this plant, and with the slag so high in lime and so low in iron, there is little danger of cutting. Indeed, the hearth is at times more inclined to chill than to cut, and the trap-spout has of late been raised so as to retain

11 in. of matte permanently in the crucible, instead of 9 in. as was the former practice; this avoids crusts in the region of the crucible.

The powerful melted stream flows into circular settlers, or forehearths — three for the large furnace, two for the smaller furnaces. These settlers are formed of half-inch (0.012 m.) steel plates, and are 16 ft. in diameter and 5 ft. in height. They are lined with 18 in. of silica brick, and have a filling of four inches of crushed quartz between the shell and the brick lining. The bottom is formed of 12-in. silica brick laid on end, and is covered

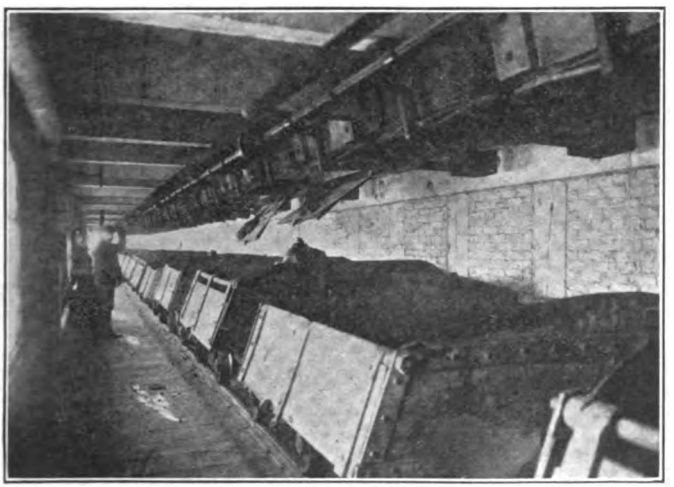


FIG. 58. — Washoe smelter. Charging train loading briquettes

with a thick layer of crushed quartz. A perforated sprinkler-pipe surrounds the outside shell in case the settler is running too hot.

The stream of melted products from the furnace enters the settler through a hole in the slag roof, the overflow slag discharging continuously over an uncooled cast-iron spout bolted to the upper edge of the shell opposite the trap-spout. If the slag runs a little higher than usual in lime or silica, the settler tends to chill and fill up, and is restored to its normal condition by correcting the composition of the slag, FeO being the great solvent. A settler lasts for several years, but requires repairs about every three months at the front side, and in the region of the matte tapping-hole. This repairing takes about 20 minutes to effect, and is accomplished

by allowing the settler to become nearly filled with matte, closing the trap-spout, and then tapping out the settler clean. A considerable quantity of crushed quartz is then thrown in through the opening in the roof, and is tamped rapidly into place by means of bars shaped for the purpose.

The settler is provided with two tap-holes for matte, which consist of massive uncooled cast-iron blocks, four inches thick. These last for about four weeks, and can be replaced without stopping the furnace.

The matte is tapped, as required, into a 9-ton steel ladle which runs on a track situated below the blast-furnaces and above the converters.

The continuous stream of slag is granulated by a jet of water from a six-inch pipe, flattened at the end, the water being the overflow from the settling-ponds of the concentrator.

The matte runs about 46 per cent in copper, and the slag only about 0.3 per cent. The labor upon the tapping floor, per shift, consists of a tapper and two helpers for each settler — or six men for the smaller furnaces, and nine for the large furnace.

The tonnage per square foot hearth area is somewhat less for the central sections than for those at the end, the former smelting about as much per sq. ft. as the old 15-ft. furnaces, while the end sections have an increased capacity. No uniform difference in the degree of concentration between center and end sections can be determined.

The method of tapping these large settlers is always of interest to the practical man. The old-time chiseling-out of the tap-hole is seldom seen in modern furnace practice. The perforated iron tap-block offers a specific and circumscribed point of attack, and the clay with which it is plugged is prevented from becoming a formidable barrier by means which I will describe in detail. The tapper determines the amount of matte that the settler contains by lowering into it an iron rod on which, when withdrawn, it is easy to see the junction-line of slag and matte.

In tapping, the iron bar which always remains in the tap-hole (penetrating the clay plug) is knocked out in the customary manner, and the matte flows through a clay-lined runner into the 9-ton ladle on the track below. This ladle is lined with crushed silicious ore mixed with sufficient concentrator slimes to impart adhesiveness, and is dried with slag.

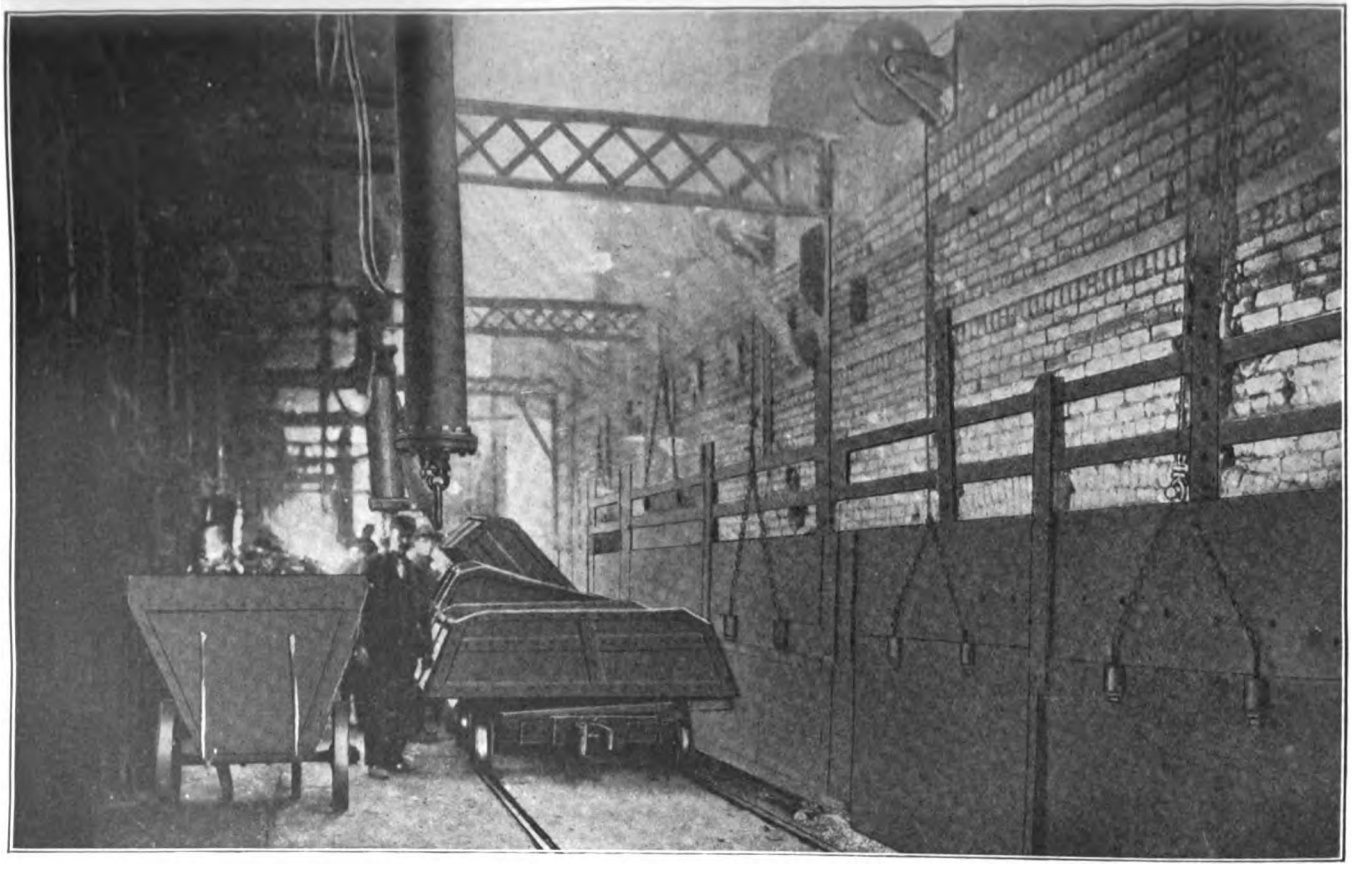


FIG. 59. — Washoe smelter. Blast-furnace charging-floor

As the matte gives off dense fumes of ZnO, and as it is impossible to plug an invisible tap-hole while it is still discharging a powerful stream of matte, a sheet-iron hood is lowered over the tap-hole, and an air-blast employed to keep it clear of fumes. A clay plug can then be introduced in the usual manner, and a warmed pointed iron bar is driven through the soft plug until its point reaches the liquid interior of the forehearth. The point of the bar is, of course, slowly dissolved by the matte, so that it is necessary to drive it in a little from time to time. If this is neglected and the bar becomes frozen in the hole, it is necessary to remove it entirely and to chisel through the hardened clay until the softening of the material indicates danger, when the pointed bar is driven in as before.

The temperature in these settlers is very high, which fact, in connection with the earthy slag, accounts for the extremely small loss in copper, the assay of the slag in that metal sometimes running down to 0.2 per cent for a considerable time, while the matte contains 46 to 47 per cent copper.

The following figures are from the smelter records: temperature of molten stream leaving trap-spout of blast-furnace, 1325 deg. C.; temperature of matte as tapped from settler, 1240 deg. C.; temperature of slag at overflow, 1290 deg. C.

The average amount of charge (everything excepting coke) smelted per sq. ft. hearth area in the Washoe blast-furnaces for a period covering several months was 5.968 tons (58.25 m. t. per sq. m.). Considering the low-iron high-lime slag, and recollecting also that it takes much longer to burn sulphides than to burn coke, this is a fair, though not large, duty.

The amount of air used is very large and, on account of the low ore-column, is probably not utilized to the best advantage.

The blowing plant for these three large furnaces comprises 12 Connersville rotary-piston blowers, of which the older ones are belt-driven and displace 300 cu. ft. free air per revolution, while the later ones are operated electrically and displace 320 cu. ft. per revolution. It is found better to use gearing only at one end of the blower shafts, and to save curves in the blast pipe by taking in the air at the bottom of the blower and discharging it at the top. The average pressure is slightly above 40 oz. per square inch.

Ten of these blowers are usually in action, and about

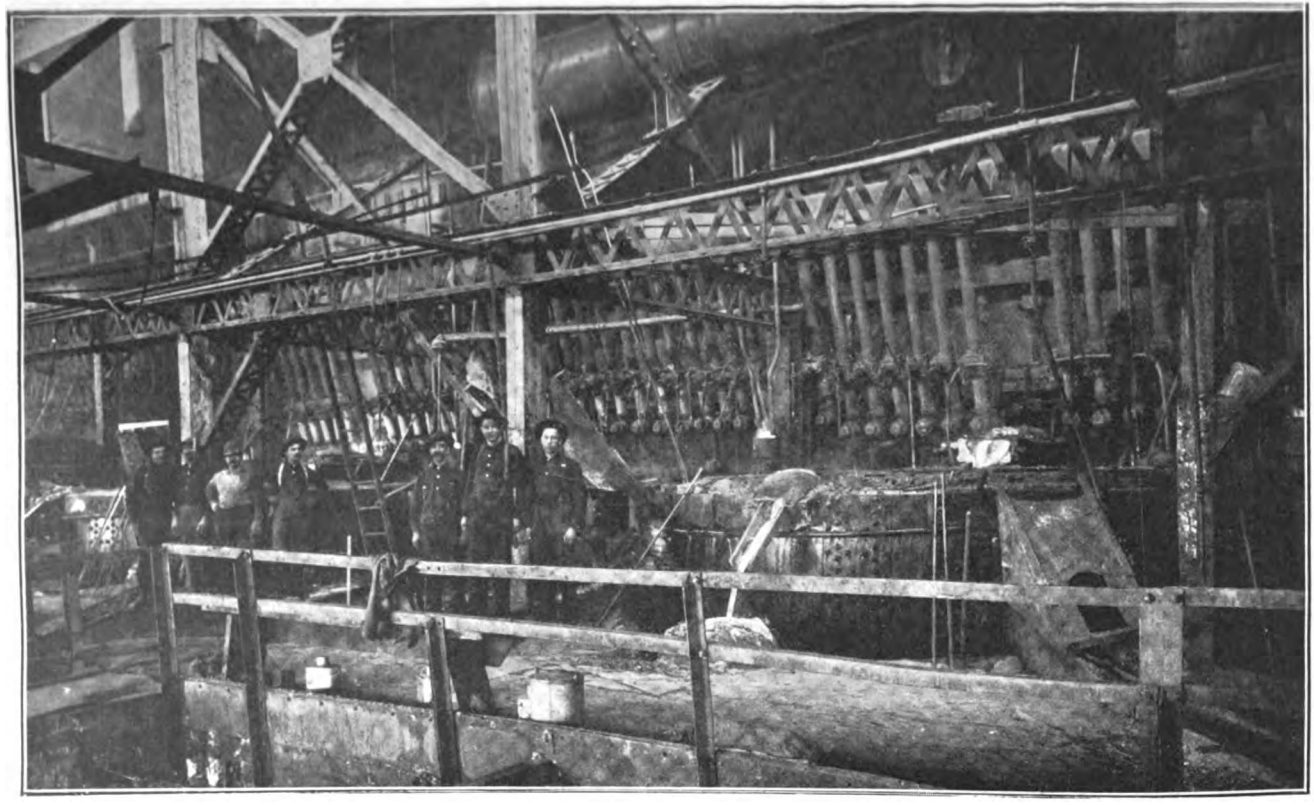


FIG. 60. — Washoe smelter. Large blast-furnace

[illegible] cu. ft. of free air is blown into the furnaces each 24 hours, which is about 60,000 cu. ft. free air per ton of charge smelted.

The managing of a partial pyrite process, even where the best technical skill is available, is a very different matter from the managing of a coke-burning furnace on roasted ore.

In the latter case, the metallurgist dictates his terms and the furnace complies with them, provided they are in any way reasonable. Being independent of heat-producing reactions in the charge itself and controlling the situation by the overpowering influence of abundant coke, he can force the process to conform to his views over wide limits. But in partial pyrite smelting (and the nearer it approaches true pyrite smelting, the more strongly do these remarks apply), he has to follow rather than lead.

To be sure, he has the control of three factors: blast, silica, and coke — but he cannot exert this control in an unfettered manner. As soon as he establishes two of these factors, he loses control over the third, and must follow where the process leads. Therefore, any attempt at modifying an established charge is more or less of an experiment, and is determined by cautious trial rather than by previous calculation.

One of the main difficulties encountered at smaller plants, when attempting to cut down their coke and smelt their sulphide ores without roasting, has always been the lack of blast. It requires an astonishing excess of wind to have any oxygen left after burning the coke, and the available supply is often misjudged by relying upon the manometer and the blower revolutions, and assuming that the furnace is receiving a corresponding amount of blast. The only air that counts is the air that actually enters the furnace.

Heavy and unsuspected losses of air are experienced even at our best managed plants. Only a short time ago, the blast-furnace superintendent of one of our principal American smelters informed me that, since making absolutely tight joints between bustle-pipes and tuyeres, he had been able to shut off one-sixth of his blowers.

It is a curious fact that, even with the high temperature attained in the Washoe blast-furnaces, a very slight increase in their silica content above the normal 39 or 40 per cent has a serious effect upon the fusion. The metallurgist in charge informs me that 42 per cent silica is considered the danger line, and that such

a slag cannot be made for any length of time without showing its tendency to chill at the tuyeres and in the settlers.

It might be supposed that, where so narrow a margin of safety exists, any slight fluctuations in the oxidizing effect of the furnace — and consequently in the FeO content of the slag — would react so seriously upon the process as to be a constant menace. The situation, however, is not so dangerous as it appears at first sight, for the percentage of sulphur and iron in the charge is relatively so small that considerable variations in the grade of the matte may occur without any serious diminution of the FeO in the slag. As pointed out by Offerhaus, if the furnace were producing a 45 per cent matte, accompanied by a slag containing 18.6 per cent FeO and 40.6 per cent SiO_2, and some irregularity should affect the oxidizing power of the furnace so that the matte suddenly ran down to 40 per cent copper, the resulting slag would only go up in SiO_2 to 41.1 per cent, and down in FeO to 17.7 per cent.

The lowest FeO in the slag for any considerable time was 17 per cent. Fifteen per cent was tried for a short time, but made trouble by chilling. The 17 per cent FeO slag contained 30 per cent CaO and 41.5 per cent SiO_2. It was very low in copper, but difficult to manage, and inclined to chill rapidly in the settler.

The cost of smelting, per ton of ore, in the Anaconda blast-furnaces is rendered high by the silicious character of the charge, which requires an unusual addition of limestone. Much of the ore-charge consists of granular concentrates and of still finer concentrates and screened ore in the shape of briquettes. It is believed to be about $2.50 per ton of ore, but this is not from any official statement.

As the blast-furnaces at the Washoe smelter are unsuited to ordinary conditions, owing to their great size, I will introduce plans, with a brief description, of the eight modern blast-furnaces built at Cananea, Mexico, in 1908, to replace the old furnaces.[1]

I have selected this plant for description because it was designed for the production of a much lower grade matte (about 32 per cent copper) than is the case with most of the large works in this coun-

[1] These plans were published in the *Engineering and Mining Journal* of April 25, 1908, by Charles F. Shelby, Superintendent of Reduction Division, Cananea Consolidated Copper Company. I make free use of Mr. Shelby's description, as well as of later information concerning the same plant by Courtenay DeKalb, in the *Mining and Scientific Press*, of July 2, 1910, p. 9.

try, and thus offers an instructive contrast. More recently, it has been found wise to increase the oxidizing action of the furnace, and the matte at present runs about 37 per cent copper.

The type of Cananea furnace which is shown in the accompanying illustrations has an inside measurement at the tuyeres of 48 in. by 210 in., or a hearth area of 70 square feet. The height of the smelting column is 10 ft. 3 in.[1]

The feeding mechanism, indicated in Figs. 61 and 62, is designed to operate in harmony with the elaborate bedding system for ores which has been installed at Cananea. The already fluxed charge from the ore-beds is delivered at each furnace by a 20-in. conveyor-belt, and an automatic tripper to a 75-ton storage bin. From this bin the charges are withdrawn as needed at each side of a furnace to five one-ton hoppers. These hoppers are so designed that the charge can be dropped direct into the furnace, or may be dumped upon the feeding-floor, and charged by hand if the furnace needs special humoring.

The water-jackets are of unusual size, the side jackets being 12 ft. 6 in. in height. The water-space is 5 in. wide, and the inner sheets are of $\frac{1}{2}$ in. flange-steel, the outer, of $\frac{3}{8}$ inch.[2]

Each furnace has 36 tuyeres of five inches diameter. The total quantity of jacket-water used is 340 gallons (1287 lit.) per minute, the water entering the jackets at a temperature of 21 deg. C., and leaving at 40 deg. C. If water were scarce, it is evident that a much less quantity would suffice.

The lower part of the furnace-hood is built up of one-quarter inch plates, spaced nine inches apart, thus forming an air-jacket around the furnace, with open top and bottom. This prevents accumulations of slag, or other substances, which are sometimes forcibly ejected from blow-holes in the charge, and also protects the workmen from radiated heat. The goose-necks, of 6 ft. diameter, would better have been larger, as shown by experience.

Owing to the low grade of the matte, the jackets extend further below the tuyeres than is customary, and two tap-holes are provided through them to tap the crucible dry when necessary.

The breast-jacket, shown in Fig. 64, is an interesting feature, consisting of a solid cast-iron block which is cast around a looped

[1] One foot (12 inches) = 0.012 m.

[2] I am informed that during a term of service ranging from two to three years, not one of these 128 long water-jackets has been replaced.

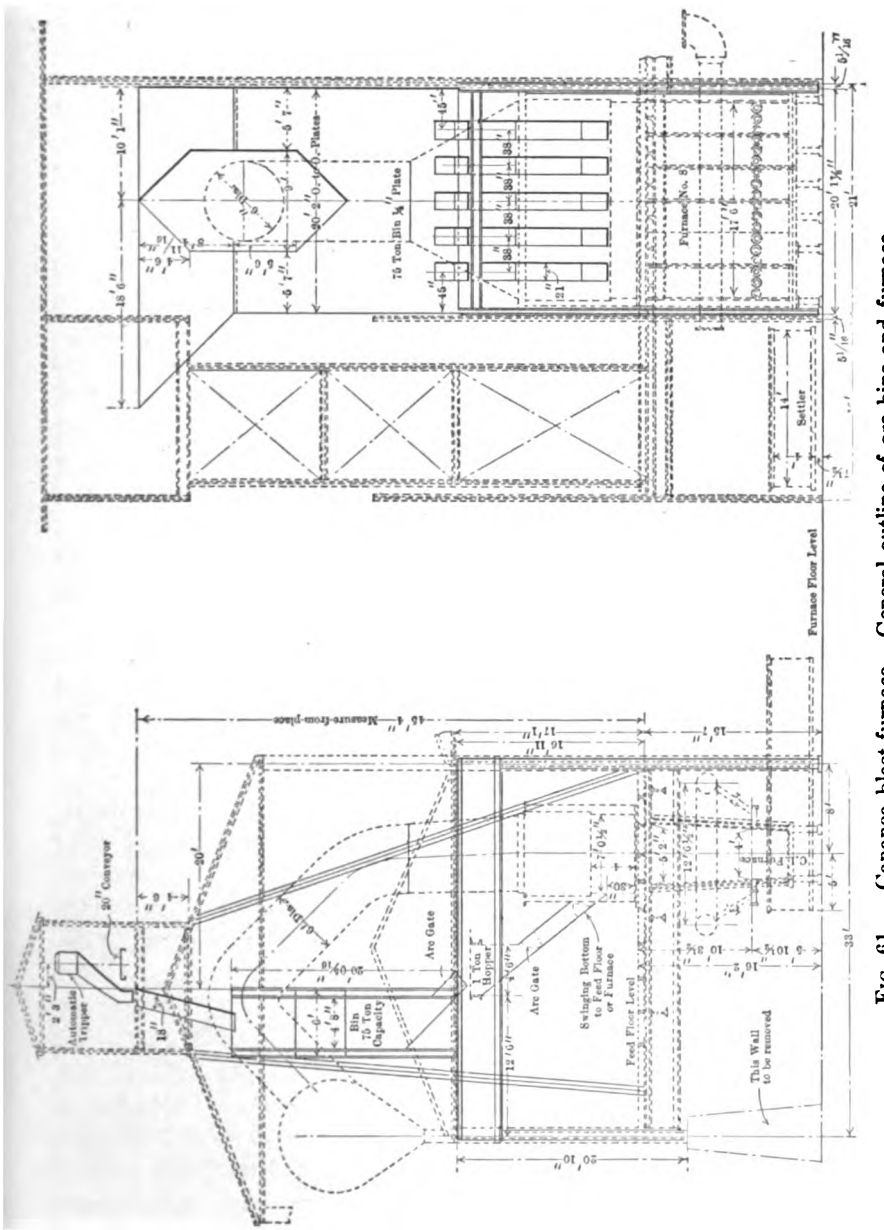


FIG. 61. — Cananeа blast furnace. General outline of ore bins and furnace

1½ in. extra heavy iron pipe. Shelby finds that if the pipe-coils in it are properly looped in, with the principle in view of cooling the surface of the jacket where it is most exposed to heat, it will last almost indefinitely. In case of carelessness, so that the upper part of the tap-hole is burned away to the pipes, no serious explosion will occur, as a small hole will first be burned in the pipe, and will be shown immediately by the boiling of the melted products in the spout. He regards this safer than the ordinary copper jacket with water space.

Fig. 67 shows the details of a tuyere. The main portion is fastened, by means of a finished flange, to a companion flange already bolted to the jacket, and makes its union with the blast connection by means of a rough ball-and-socket joint which is provided with a groove for an asbestos gasket; by adjusting the bolts which fasten these two parts together, a perfect alignment of the blast connections can be maintained. The lower part of the tuyere is provided with a slag-escape hole stopped with a wooden plug, which Shelby prefers to more elaborate methods. The cap of the tuyere is seated to the tuyere itself by means of an air-tight ground joint.

In Figs. 66 and 68 are shown the details of the forehearth or settler, as well as the method of lining the same for low-grade matte, which has proved satisfactory. Two blast-furnaces discharge into one settler and, of the total four settlers which have been in use for 2½ years, one has had a new lining, and two others have had some repairs on the lining. I describe at some length Shelby's method of lining these settlers, as the subject is an important one in practice.

The bottom is put in with 10 inches of refractory fireclay brick, the upper layer of which is replaced by one layer of chrome brick directly underneath the spouts and around the tap-holes. The sides are lined with nine inches of chrome brick all the way up, but a 12-in. space is left between this lining and the shell of the settler, to be filled with a mixture of crushed quartz and clay such as is used in lining converters. Great importance is attached to this so-called "converter-lining" as chrome brick expand considerably, and are also good conductors of heat, and the lining corrects both of these faults. While Shelby prefers magnesite for converters, he finds the chrome brick more suitable for settler linings, and the Cananea experience certainly speaks well for the latter

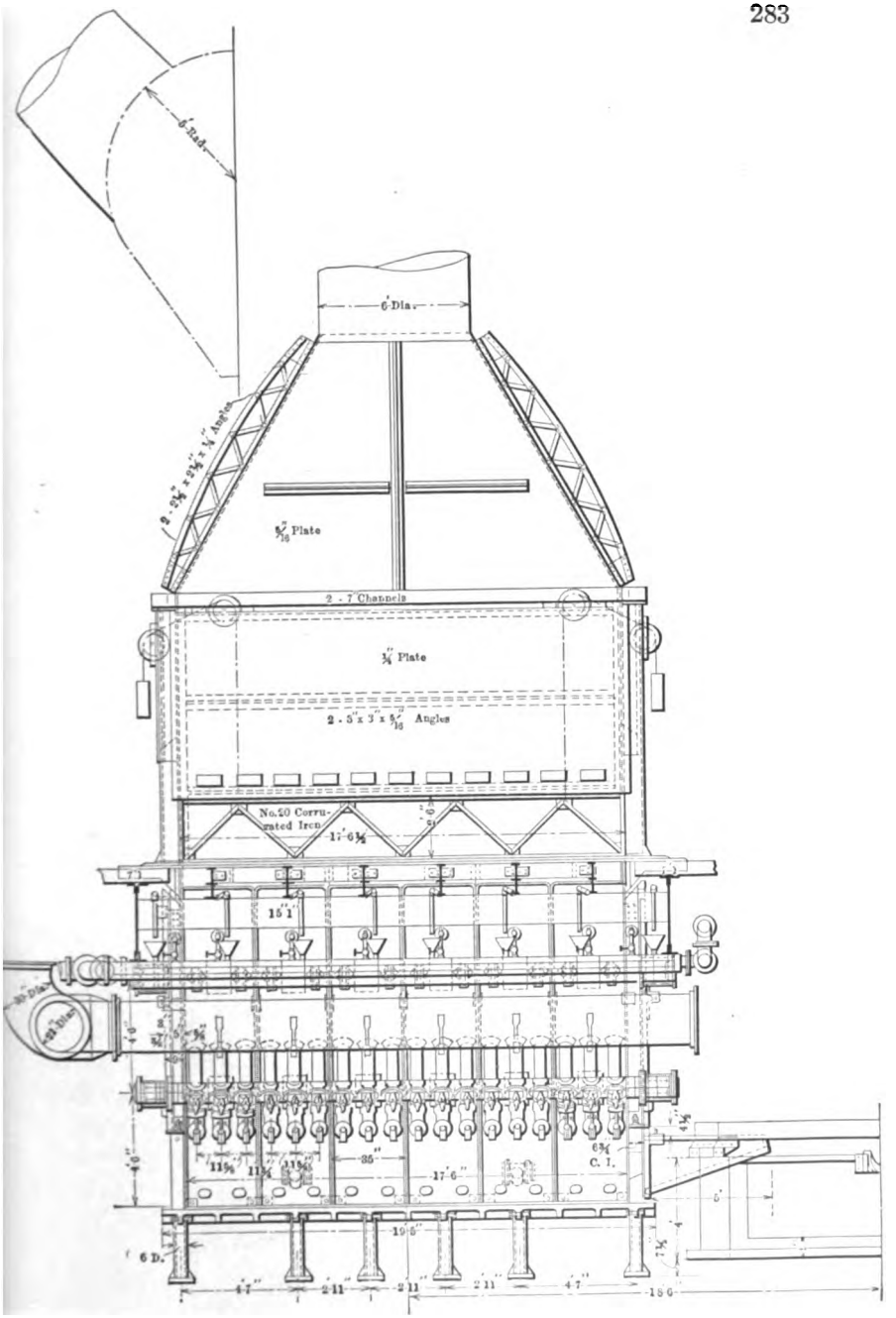


FIG. 62. — Cananea blast furnace — side

material. As a precaution against overheating, a spray-pipe may also be used upon the outer shell.

These furnaces were run originally at a rate of about 450 tons charge per day, or 6.43 tons per sq. ft. hearth area. It is now found more economical to run them at only about 280 tons per day, or four tons per sq. ft., which is unusually slow for this kind of smelting, but is necessary to diminish the flue-dust.

A blast pressure of only about 16 oz. is employed, and 12,000 cu. ft. free air is supposed to enter each furnace per minute. The average temperature of the air is about 18 deg. C.

The flue-dust amounts to eight per cent of the gross weight of the charge, there being a large proportion of concentrates. The dust is settled in a brick chamber, with roof of rails and concrete, 32½ ft. high and 60 by 181 ft. in area. The flue to the stack is 20 ft. wide by 19½ ft. high, and there is a second, smaller dust chamber at the base of the stack. The temperature of the gases at this point is 230 deg. C.

The charge is made up of half lump ore from the company's mines and half concentrates, and requires about 10 per cent of coke. A small amount of limestone flux is used, seldom exceeding eight per cent.

The labor required for the six furnaces now in blast is one foreman and two chargers per shift, besides one feeder, one helper, and one coke-wheeler per furnace per shift. The small amount of labor is due to the fact that, as already intimated, the ores are bedded, ready for the furnace, in an adjoining yard in beds of about 8000 tons, and conveyed direct, by belt conveyors, to the small charging-hoppers immediately above the tunnel-head of the furnaces. This bedding and conveying system has been found highly economical and successful in the dry climate of Cananea.

The charge contains 17.11 per cent sulphur, 26.5 per cent iron, and 6.39 per cent copper, this being figured on the gross weight.

The slag approaches a bisilicate, its average composition for a considerable period being

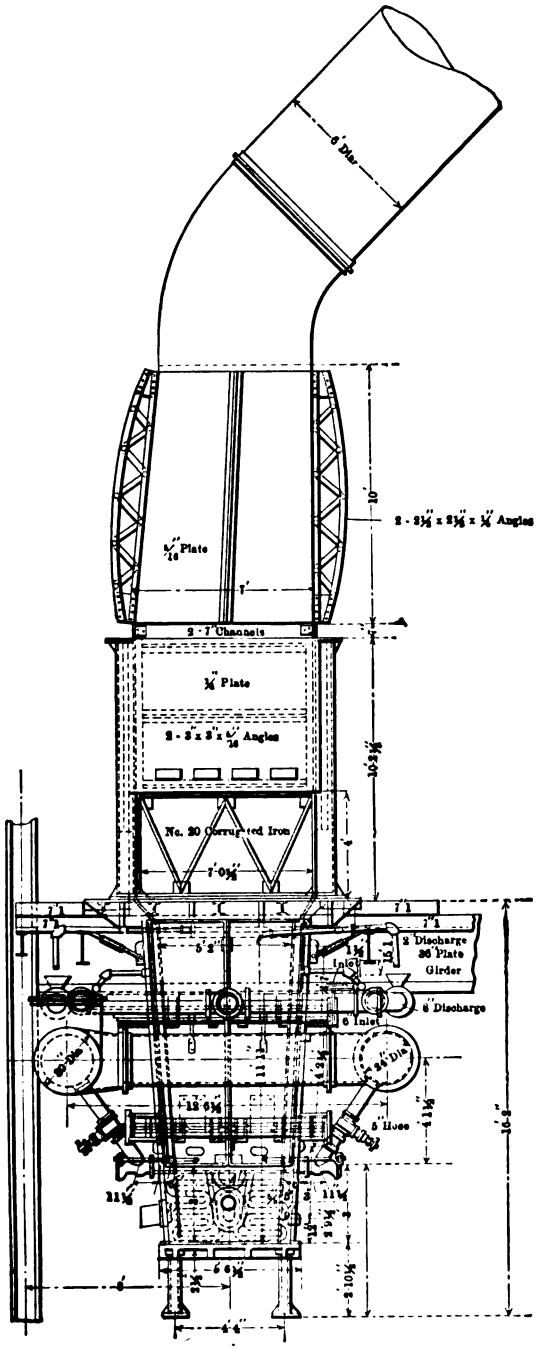


FIG. 62*a*. — Cananea blast-furnace — end

SiO_2	39.00	per	cent
FeO	36.30	"	"
CaO	11.10	"	"
Al_2O_3	9.20	"	"
ZnO	1.50	"	"
MnO	0.53	"	"
MgO	0.48	"	"
Cu	0.40	"	"
S	0.70	"	"
	99.21	"	"

The matte-fall is about 18 per cent. The matte has the following composition, with undetermined, but very small, proportions of arsenic and antimony:

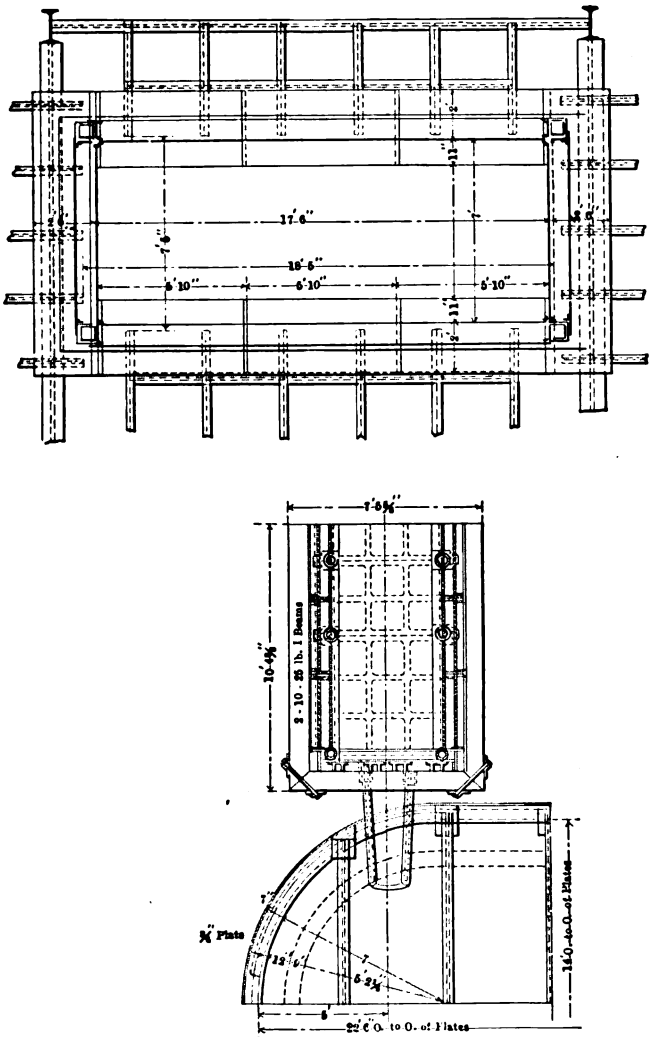
Cu	37.4	per	cent
Fe	34.8	"	"
S	26.1	"	"
SiO_2	0.5	"	"
	98.8	"	"

The coke contains

moisture	0.65	per	cent
fixed carbon	78.79	"	"
volatile matter	2.02	"	"
ash	18.40	"	"
	99.86	"	"

The process at Cananea is notable for the unusual proportion of finely-divided sulphide material in the charge, and the consequent necessity for a light blast and slow running. Both of these conditions favor roasting rather than bessemerizing, and tend to scatter the potential heat of the charge rather than to concentrate it. Although, at Cananea, the finest of the concentrates are roasted, and smelted with flue-dust in a reverberatory, there remains still so much fine material for the blast-furnaces that this portion of the plant is hampered seriously. This is fully realized by the management, and means are now under consideration to obviate the difficulty, as will be noted when discussing the treatment of fine material.

Barite, or heavy spar ($BaSO_4$) is a mineral dreaded by all experienced blast-furnace smelters when it occurs in any considerable quantity. Unaltered by any roasting process, it is


17'6"
16'6"
5'10"
5'10"
5'10"
1'6"
1'6"
1'6

reduced in ordinary blast-furnace work to BaS, which is a heavy refractory material, seldom completely liquid, too heavy for the slag, too light for the matte, and consequently dividing itself between both of these substances, with most damaging results.

In the reverberatory furnace, it is decomposed in the presence of silica, and combines with that acid to form slag, barium oxide not decreasing the fusibility of ordinary slags. In true pyrite smelting, the same end result is attained as in the reverberatory, and the decomposition of the heavy spar is satisfactory and complete.

In partial pyrite smelting, the success of this decomposition of the heavy spar depends upon whether the metallurgist is able to exert sufficient oxidizing influence at that stage of the process where the temperature is high enough to awaken thoroughly the affinity between free silica and barium oxide. Too much coke at the focus will, of course, destroy the oxidizing influence at just the point where it is needed to decompose the barite, while too little coke means too low a temperature for the reaction, and a benumbing of the entire process. It is at just this point that the value of a hot blast becomes peculiarly apparent. In order to steer between a reducing atmosphere on the one hand and a chilled furnace on the other, the metallurgist has to trim his course with more exactitude than is possible in an apparatus like the blast-furnace. If he could only raise his temperature a few score degrees without having to burn carbon to do it he would have a clear course and an ample margin of safety. This trifle of extra heat which is needed to make things comfortable can be gained, free of any reducing influences, by preheating the air blast, and it is for just this class of cases that there can be no question as to the great advantages to be derived from the practice.

An illustration of the successful treatment of heavy barite ores in the partial pyrite furnace seems advisable and, as it also permits the examination of the smelting of the identical ore with and without preheated blast, it will serve a double purpose.[1]

[1] The following brief description of operations at the Tyee Copper Company's smelter on Vancouver Island, British Columbia, is based mainly upon personal communications from Mr. Thomas Kiddie, lately general manager of the company; upon a descriptive paper in the *British Columbian Mining Record* of July, 1905; upon the paper by George W. Maynard in the *Engineering and Mining Journal* of Nov. 6, 1909; and upon a variety of other data for which I am unable to make acknowledgment.

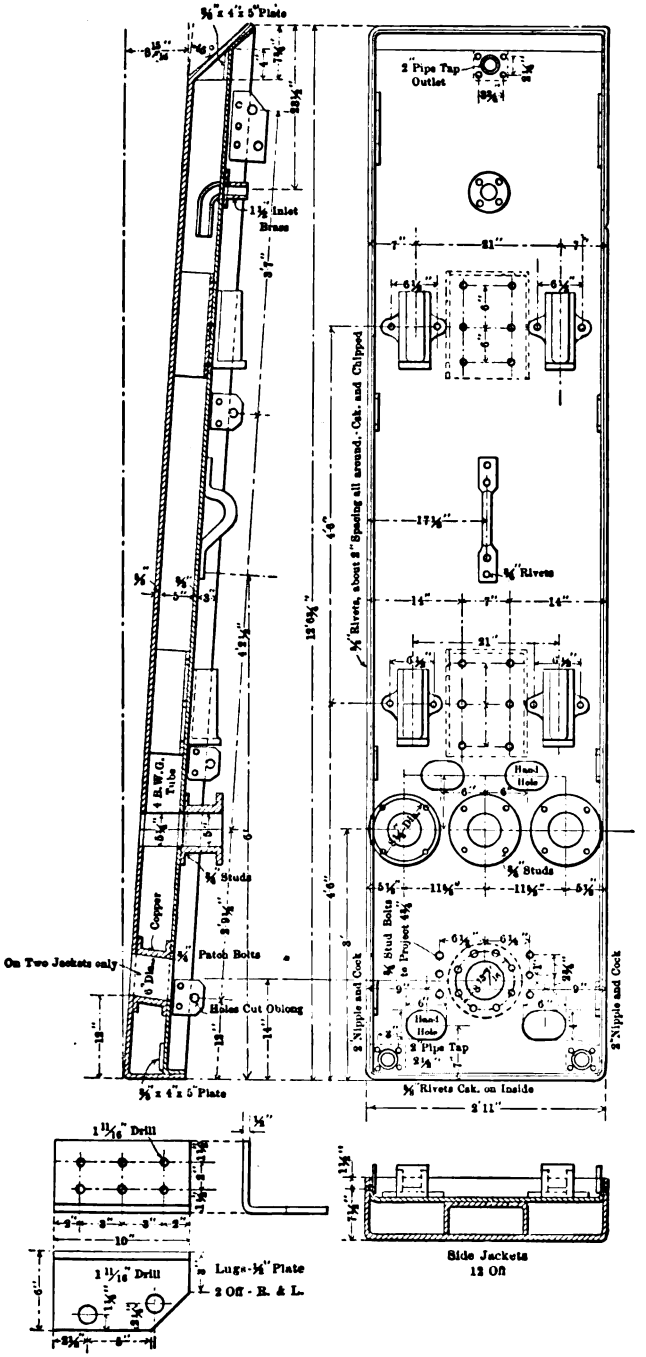


FIG. 63. — Cananea blast-furnace — details of jackets

The most interesting feature of the Tyee ore is its unusually high proportion of barite. It contained also enough zinc blende to be unpleasant, and was low in silica. The ore at the time to which most of the following description applies had, approximately, the following composition:

Cu	4.56	per cent
Fe	11.94	" "
Zn	6.60	" "
SiO_2	13.50	" "
Al_2O_3	3.95	" "
$BaSO_4$	37.30	" "
CaO	2.20	" "
S (in sulphides)	16.62	" "
	96.67	" "

The ore contained per ton 2.87 oz. silver and 0.14 oz. gold.[1]

A certain proportion of these ores was roasted in heaps and, as the ore when mined contained a large proportion of fines, it was screened; the fines were made into brick with water only, dried upon a drying floor, piled into heaps, and burned in much the same manner as common red brick. These bricks burned remarkably well, yielding a black, hard, porous product containing about $3\frac{1}{2}$ per cent sulphur (apart from the barite), and with almost complete oxidation of the blende. The lump ore also burned well in heaps, although still retaining about 7 per cent sulphur in the form of sulphides.

The burned ore (proportional average of lumps and bricks), for a period of three months, had the following composition:

FeO	17.60
ZnO	8.90
Al_2O_3	3.61
CaO	3.96
$BaSO_4$	39.74
SiO_2	11.10
Cu	4.60
S (apart from barite)	7.94
	97.45

As the company could obtain a cheap supply of an aluminous schist carrying a little disseminated copper, it was determined to

[1] One ounce per ton = 0.0034286 per cent.

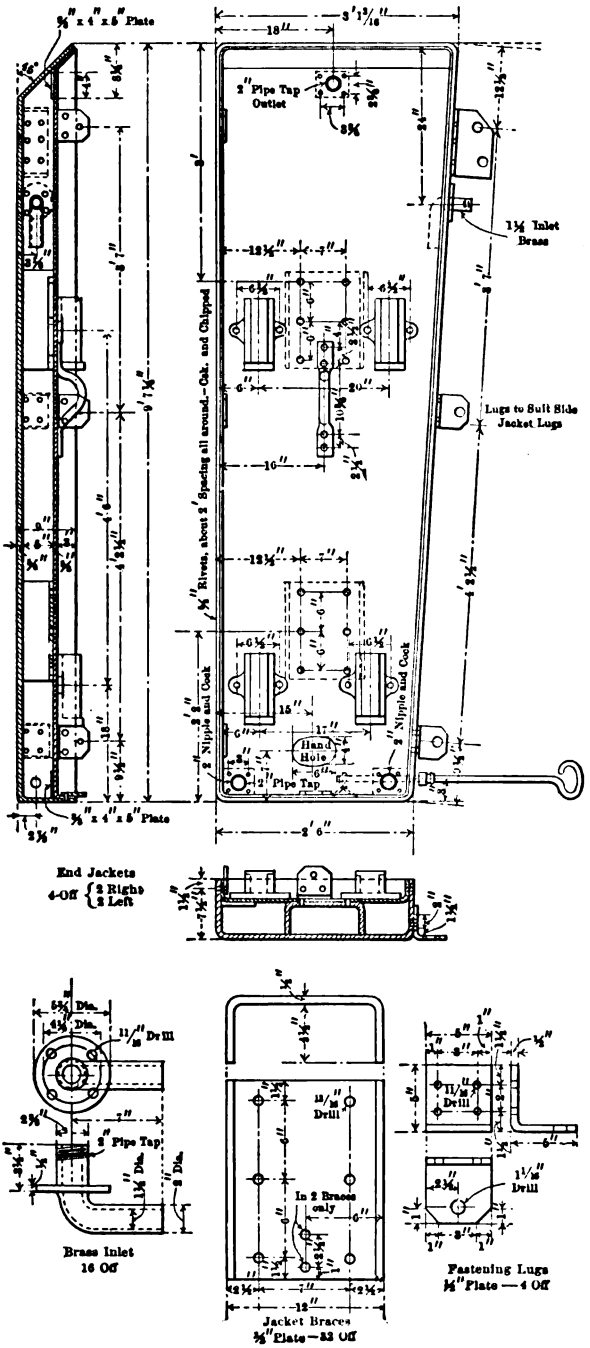


FIG. 63a. — Cananea blast-furnace — details of jackets

use this as an acid flux, hoping to form aluminate of baryta. This was certainly a pretty hard proposition and, although by no means so unsuccessful as one might suppose, made it difficult to carry out the process in exactly the manner desired. Free silica was plainly indicated, and was finally obtained in the shape of an ore from Idaho carrying 80 per cent free silica and four per cent copper in the form of carbonate.

No sulphate, so far as I am aware, can withstand the contact of free silica and an oxidizing atmosphere at a temperature of

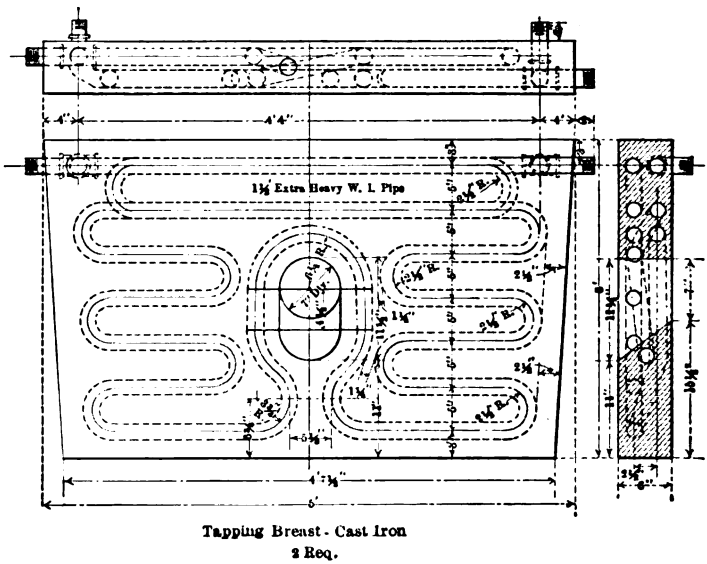


FIG. 64. — Cananea blast-furnace cast-iron tapping breast

something like 1100 deg. C., and the decomposition of the barite (and, consequently, the degree of concentration of the matte) became entirely satisfactory under these improved conditions, and with the addition of a hot blast. It was not until the blast was preheated that the proportion of coke could be cut down low enough to bring about the proper oxidizing condition in the focus.

With cold blast, it was necessary to have the mixture contain two-thirds of burned ore and one-third of raw ore, in order to produce a 42 per cent matte, while, with the heated blast, the same grade of matte was attained by the use of two-thirds raw ore and only one-third roasted ore.

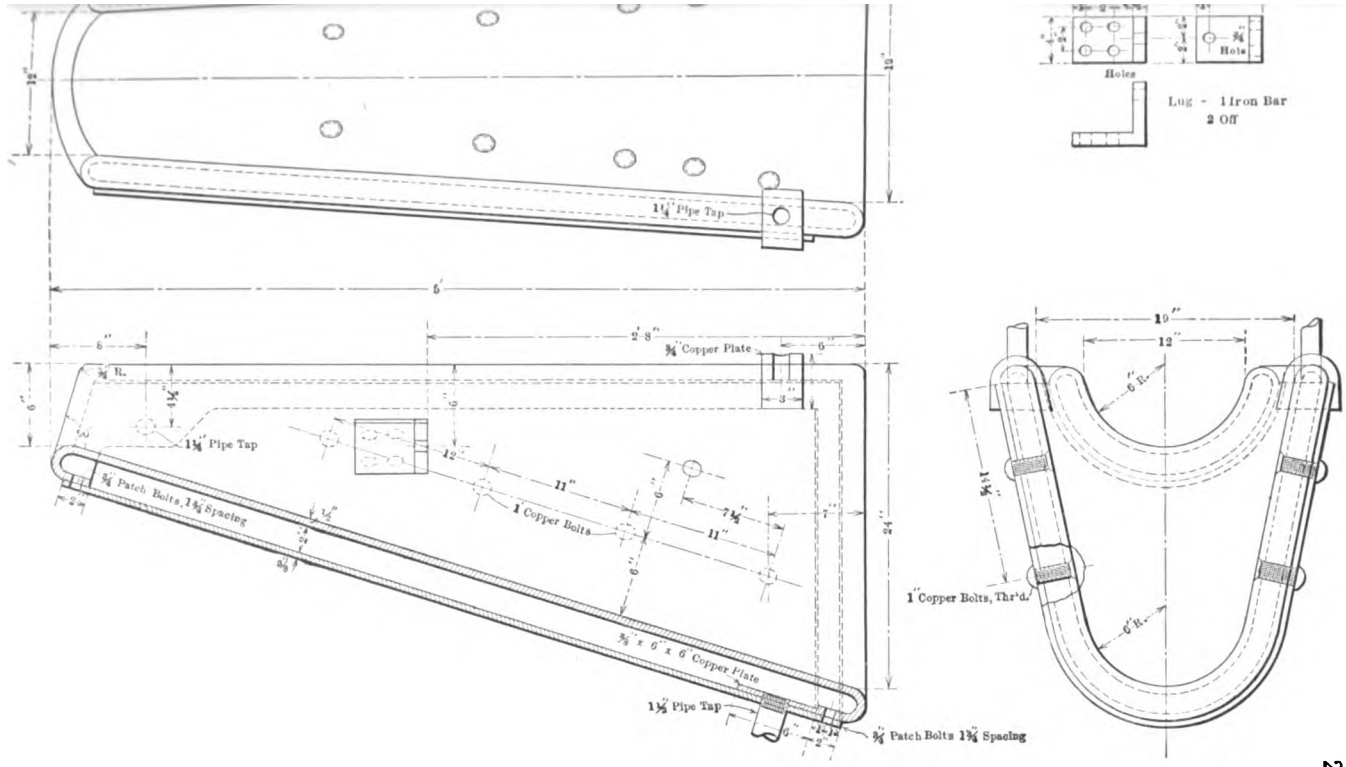


FIG. 65. — Cananea blast-furnace. Water-jacketed trap-spout

Mr. Kiddie writes me that his experience taught him that the four vital requirements for this kind of work are:

1. Low proportion of coke.
2. Great volume of blast.
3. Free silica.
4. Low ore-column.

And I may add that, in order to lower his coke to the smallest possible proportion, he needs a hot blast, so as not to steal the scant heat in front of the tuyeres.

A very moderate heating of the blast produced at the Tyee smelter, as well as at many other furnaces well known to American metallurgists, a remarkable saving in coke and increase in oxidizing power, as well as in ease in managing the operation. The blast at the Tyee furnace was heated by a device of which the effective details were originated by Mr. Kiddie, consisting of a pipe within the main flue of the furnace, arranged in such a manner as to present a maximum of absorbing surface to the retarded waste gases. The temperature at which these gases left the furnace was comparatively low, so that the blast was warmed only to about 200 deg. C.; yet even this slight aid was accompanied by a reduction of 25 per cent of the amount of coke used, as shown in the annual report of the company.[1]

Later practice has resulted in using considerable additions of custom ores higher in silica and alumina, and in producing a richer matte and cleaner and more acid slag. A late typical analysis of the slag is as follows:

Cu	0.38	per cent
FeO	19.42	" "
SiO_2	37.20	" "
BaO	24.11	" "
CaO	3.50	" "
Al_2O_3	9.16	" "
ZnO	5.12	" "
	98.89	" "

[1] I will again call attention to the fact that the great benefit experienced from preheating the blast in this particular form of partial pyrite smelting does not apply to true pyrite smelting. In the latter process, there is abundant fuel present in the shape of sulphides, and the way to develop this potential heat most rapidly and to the best advantage is to blow in great volumes of our nearest feasible approach to concentrated oxygen — which, at present, is cold air. High ore-column, powerful blast, cold air, and a fierce bessemerizing action are the conditions for true pyrite smelting, so far as we yet know them.

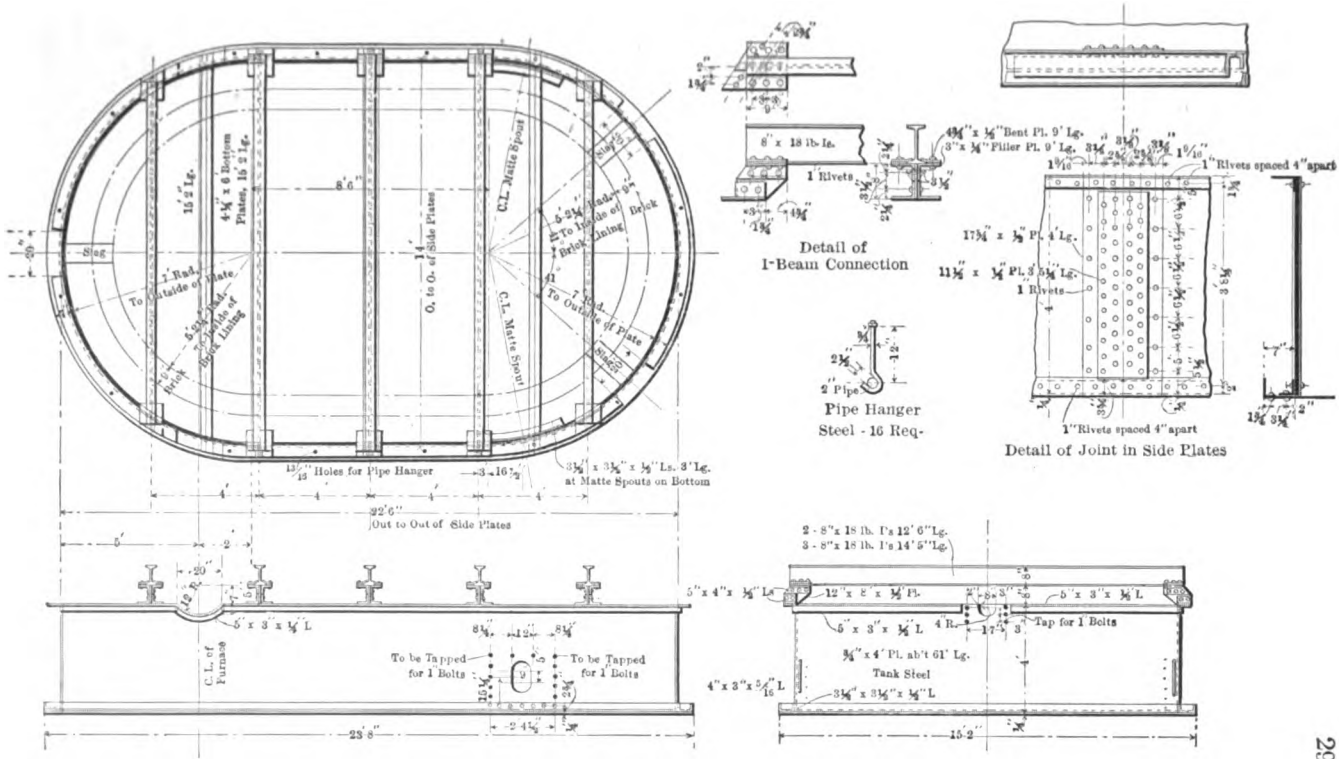


FIG. 66. — Cananea blast-furnace — plate work of settler

The matte from this charge is stated to have about the following composition, although I do not offer it as an analysis:

Cu_2S	50.60
FeS	32.01
BaS	4.20
ZnS	11.04
	98.85

It contained per ton 32.6 oz. silver and 1.67 oz. gold.

There is no trouble in accounting for this extensive decomposition of heavy spar in a reducing atmosphere, and in the presence of free silica and a metal, or a metallic sulphide. Kiddie, finding only traces of SO_3 in the furnace gases, suggests that the barite does not undergo direct decomposition in the manner indicated, but is reduced to BaS, as in coke smelting, and then behaves like FeS. While it is evident, from an examination of the matte, that a small proportion of the $BaSO_4$ is reduced in this manner, Schweder's careful investigations on the behavior of barite at elevated temperatures offer the straightforward and probable reaction

$$BaSO_4 + 2SiO_2 + Fe = BaSiO_3 + FeSiO_3 + SO_2.$$

This shows that the liberated sulphur trioxide of the barite splits into SO_2 and O, the latter oxidizing the metal or sulphide.

The 40 by 120 in. furnace smelts about 230 tons charge per 24 hours, using 11 to 12 per cent coke, and making a concentration of 10 or 12 into one. The coke is high in ash, as follows:

fixed carbon	67.3	per	cent
volatile matter	9.77	"	"
ash	21.99	"	"
sulphur	0.94	"	"
	100.00	"	"

Superintendent Musgrave gives the specific gravity of the Tyee mattes as between 4.7 and 5.0, and that of the slags as between 3.6 and 3.8.

My purpose in these separate illustrations of modern metallurgical practice is to represent types of the application of the partial pyrite operation to various diverse conditions, where both chemical reactions and technical manipulations may offer material for study and information.

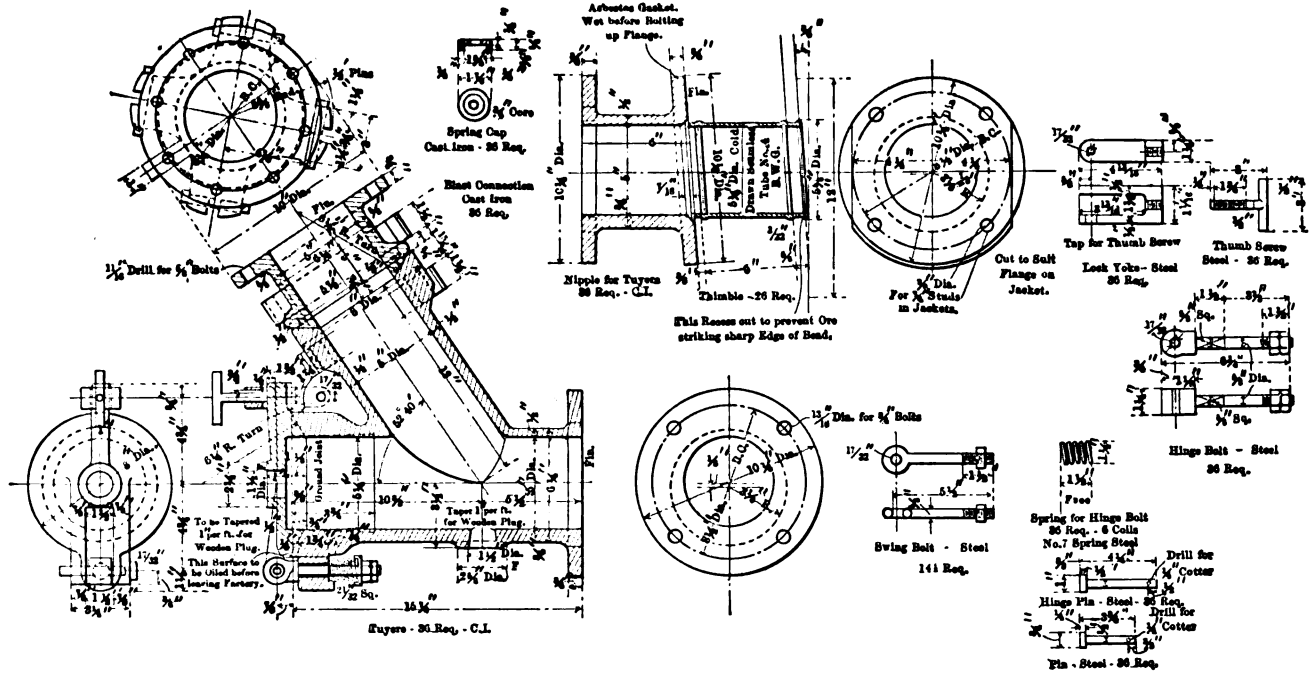

Asbestos Gasket. Wet before Bolting up Flange.
Spring Cap Cast Iron - 36 Req.
Blast Connection Cast Iron 36 Req.
Nipple for Tuyere 36 Req. - C.I.
Thimble - 36 Req.
This Recess cut to prevent Ore striking sharp Edge of Bend.
Cut to Suit Flange on Jacket.
Tap for Thumb Screw
Lock Yoke - Steel 36 Req.
Thumb Screw Steel - 36 Req.
Hinge Bolt - Steel 36 Req.
Spring for Hinge Bolt 36 Req. - 6 Coils No.7 Spring Steel
Hinge Pin - Steel - 36 Req.
Pin - Steel - 36 Req.
Swing Bolt - Steel 144 Req.
Tuyere - 36 Req. - C.I.

FIG. 67. — Cananea blast-furnace — tuyeres and details

I am selecting, so far as possible, illustrations with which I am personally familiar, in order that I can discuss them with more intelligence and authority, and am diverging more and more from the class of work where the oxidizing feature plays the prominent part and the combustion of the pyrite with its formation of FeO is the main slag-forming reaction.

In other words, I am approaching a unique variation of the process, where the pyrite in the charge is scanty, the silica high, and relief is sought by the addition of the least possible amount of limestone that will yield a slag of sufficient fusibility to flow and to abstain from chilling the tuyeres. At all of the great Butte smelters it is the custom to produce a slag low in iron and high in lime; but the Butte metallurgists avoid the precariousness of the process which I am about to describe by adding sufficient lime to bring the silica content of their slags down to about 40 per cent, thus escaping many difficulties. In the following process, economic conditions forbid the use of such an excessive quantity of limestone as would be required to reduce the silica to the Butte standard, nor would such an addition be possible from a metallurgical standpoint, as a slag so low in iron and so high in lime would not flow properly at the temperatures attainable in ordinary work.

Even as it is, a considerable amount of coke is required and, as the atmosphere of the furnace must still be of such a nature as to burn a certain proportion of the scanty sulphides to provide FeO for the slag and to effect a reasonable concentration of the matte, it is evident that — as in the preceding illustration — the coke must be kept down to the lowest possible limit and the whole process must be run perilously near to the danger line. Here again is a case where the preheating of the blast may yield benefit apparently out of all proportion to the actual thermal effect. When the focus of a furnace is just able to hold its own and has no heat at all to spare, the addition of the few units of heat conveyed by even a slightly warmed blast makes a difference that can be appreciated only by those who have experienced it. The process may be able to exist without it; but it is infinitely more comfortable with it, and usually more economical.

The type of process to which I refer is not only somewhat rare, but yields slags of so unusual a composition and of so high a flowing temperature that, if resulting only from the treatment of

small quantities of material, or from work covering limited periods of time, they might be regarded as exceptional, and as not establishing a sufficient basis for imitation. For this reason I shall confine my principal illustrations for this type of smelting to two

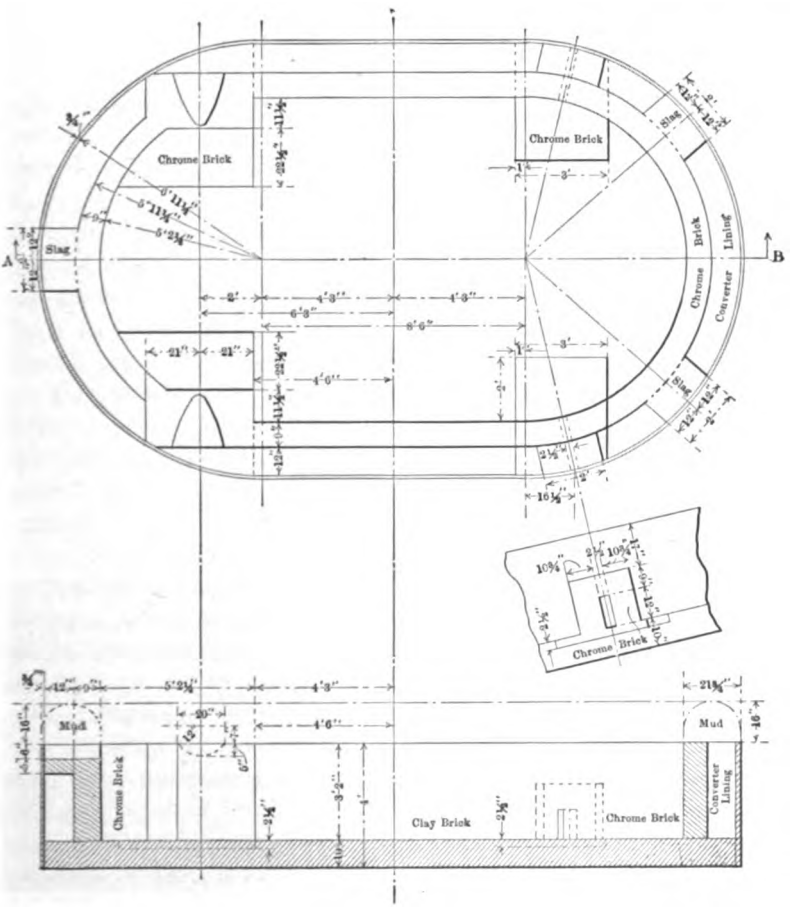


FIG. 68. — Cananea blast-furnace. Method of lining a settler

cases where these high silica-lime, low-iron slags were profitably made for a number of years on a large scale, and under adverse commercial conditions, and where their quality and chemical composition were matters of common knowledge to all American smelters who have interested themselves for this kind of work.

The two plants which are the subject of the succeeding remarks were both in the isolated gold district known as the Black Hills, in South Dakota, both making the class of slags to which I have just referred, both developed from small beginnings to large plants, and both purchasing their ores from the mines of the surrounding districts. The opening of new railway communications, the improvements in the cyanide process, and a variety of other influences incident to the settling of a new country, have gradually extinguished their narrow margin of profit per ton of ore, and they are now idle, after a number of years of active and profitable existence.

The late Franklin R. Carpenter was the pioneer in this peculiar form of pyrite smelting, although my descriptive remarks will apply more particularly to later work of much the same nature executed by Fulton and Knutzen.[1]

In the Black Hills, in 1889, there was a considerable production of a certain type of refractory gold ore for which there was no suitable market. It contained 1 to 1½ oz. gold per ton, and a few ounces of silver. The general composition of this ore was: silica, 75 per cent; iron oxide, 15 per cent; alumina, 5 per cent; lime, 3 per cent.

There were no copper or lead ores in the district, and Carpenter undertook to smelt these silicious gold ores with pyrite, and collect the gold in an iron matte of sufficient value to stand the long transportation to the nearest market. The best materials at his disposition for this purpose were (*a*) certain barren pyrites and pyrrhotites carrying 30 per cent silica; (*b*) a cheap barren dolomitic limestone; (*c*) a local, very friable, coke, with 24 per cent of ash which was 60 per cent silica, or an expensive coke from the East.

While Mansfeld, Kongsberg, Lend, and several other classic plants furnish examples of high-silica, low-iron slags that are commercially and technically successful, little that metallurgical

[1] I use freely, and without constant acknowledgment, a paper by Franklin R. Carpenter, entitled "Pyritic Smelting in the Black Hills," *Trans. Am. Inst. Ming. Engineers*, Vol. XXX, pp. 764–777, and a paper by Charles H. Fulton and Theodor Knutzen, entitled "Sulphide Smelting at the National Smelter of the Horseshoe Mining Co., Rapid City, S. D.," in Vol. XXXV, pp. 326–338, of the same *Transactions*. Also personal communications from Messrs. Carpenter and Fulton.

literature could suggest seemed to suit the situation at the Deadwood smelter, and a long series of experiments in crucibles was carried out at the South Dakota School of Mines, with a view to determining whether a high silica-lime slag could be made from the existing materials which should be sufficiently fusible for practical work, and whether the resulting iron matte would separate satisfactorily from it, carrying with it the gold and silver.

Wheat flour was mixed with the crushed materials to act as a reducing agent and, after much experimental work, it was found that a charge composed of 40 per cent of the silicious gold ore, 45 per cent of magnesian limestone, and 15 per cent of raw pyrite gave a clean fluid slag, and a button of matte carrying practically all of the gold and silver, thus obtaining the results which, according to Bloxam, are obtained from the smelting together of lime, quartz, and pyrite at Zalathna, Hungary.

On the strength of this work, a 36 in. round water-jacket furnace was erected as an experimental plant to demonstrate the commercial possibilities of the situation. The first slags were duplicates of the experimental ones, and the furnace ran as smoothly as at any subsequent period, the first carload of matte shipped to the refining works at Omaha containing 12 oz. gold and 60 oz. silver, with no copper.

In a few months a furnace 36 by 84 in. was erected; later, another furnace 120 in. long; and, eventually, a furnace 144 in. long.[1]

Subsequently, a portion of the barren pyrite in the charge was replaced by pyritic concentrates from the Homestake gold mills, but the slags were never so free from the precious metals (though reasonably clean) as were those made in the crucible experiments. Eventually, improved railway facilities made it possible to introduce a few per cent of copper into the matte by bringing copper sulphide ores from Butte, Montana, which reduced the slag losses at once.

As at Mansfeld, this earthy slag, low in iron, produced "sows" of metallic iron, gradually replacing the brickwork of the hearth, and weighing from 20 to 40 tons. Being almost impossible to break, and carrying some 20 oz. gold to the ton, they occasioned

[1] One foot (12 inches) = 0.305 m.
One ounce per ton (Au or Ag) = 0.0034286 per cent.

serious embarrassment, but were finally treated economically and satisfactorily in the reverberatory furnace which had been constructed for the working up of flue-dust. Whenever the flue of the reverberatory was undergoing repairs, advantage was taken of the opportunity to place one of these sows upon the hearth, sliding it in under the arch from the front. Three days of firing, with the addition of pyrite and copper sulphide ores, would dissolve a 40-ton sow, with the recovery of the gold in the shape of copper matte.

The nature of the materials at hand indicates the kind of slag which it was Carpenter's aim to make. As silica is the chief constituent of his payable ore, the slag must be as high in silica as possible. As iron is the most scarce and expensive constituent of his mixture, the slag must be as low in iron as is consistent with the proper working of the process. As lime is his cheapest basic material, lime must be used for a base so far as possible. Or, stating it as the practical metallurgist would be more likely to put it to himself: "I will base my calculations on my most expensive indispensable constituent, and the one of which I, therefore, must use the least amount possible, namely, iron; experience teaches us that when we try to make a slag below 20 per cent of FeO, it requires exceptional skill and experience on the part of the furnacemen, so I will start with 20 per cent of FeO. As commercial considerations require me to use the highest possible proportion of silica, and as accumulated experience shows that 47 per cent silica already approaches the danger line, I will not attempt, at the outset, to go beyond this amount. The remainder of the one hundred per cent I will make up with lime; hence, for the start, I will try a slag of 20 per cent FeO, 47 per cent silica, and 33 per cent lime, etc." I do not suggest the above as the exact proportions that would be the most suitable, but merely as the manner in which the metallurgist would weigh the proposition in his own mind.

Providing alumina is classed as a base (which I will do for convenience, if for no other reason), an examination of the reliable analyses of silicious copper slags extending through the past 50 years of metallurgical literature will show that scarcely any commercial slags are ever made that are more acid than a bisilicate. This is not due to any undue heat required to form slags higher in silica than the bisilicate. On the contrary, in certain ordinary slag types, the formation temperature lowers materially

above the bisilicate, and even the trisilicate has a lower formation temperature than the unisilicate. The trouble is not in the *formation* temperature of these high-silica slags; it is in the *flowing* temperature, which is far above the degree of heat required to form them. Hence these slags chill rapidly and, unless superheated to an extent incompatible with economical smelting, will crust the tuyeres, form accretions in the hearth, and chill the settler.

Carpenter found that a slag approaching a sesquisilicate, if alumina is figured as a base (close to a bisilicate, when figuring alumina as an acid), gave the best results under his conditions.

Lloyd, who was in charge of the furnaces for two years, says "our slag is a complex silicate, which is normally about half way between sesquisilicate and bisilicate. The typical slag should assay about as follows:

SiO_2	48.5	per	cent
FeO	16.0	"	"
CaO	18.0	"	"
MgO	12.0	"	"
Al_2O_3	3.5	"	"
RO bases	2.0	"	"
	100.00	"	"

" On this slag we have made our best runs. It is a peculiarly pretty slag of glassy luster when chilled suddenly, but of grayish-brown stony appearance when cooled slowly, and crystallizes in needle-shaped forms (actinolite?) In case of FeO becoming low, the proper course to follow is to increase the silica. In this manner we have made, with good results, slags of 53.8 per cent silica, but have never been able to make any other type of slag that will drive as fast as the one given above. At about 44 SiO_2, the Al_2O_3 appears to act as an acid, and makes a very bad slag. The type indicated in the above analysis is quite fluid, does not chill readily, and can be drawn into a fine thread. By observing these threads for thickness, one can form a very close estimate of the iron content of the slag."

Later, A. R. Carpenter, after ten months' experience with these furnaces, found that he could do his best and most rapid work on a slag approximating:

SiO_2	51.4
FeO	12.2
CaO & MgO	30.0
and probably	
Al_2O_3	3.5
RO bases	2.9
	100.00

With this slag, he was smelting about 200 tons charge (one-half being limestone) per 24 hours in a furnace having a hearth 36 by 144 in., which is 5.55 tons charge per sq. ft. hearth area (54.16 m. t. per sq. met.). Although it is true that something like one-fourth of the entire burden consists of CO_2, this is tolerably rapid smelting for this kind of work.

Dr. Carpenter informed me that it took about 12 to 14 per cent of good coke (18 per cent of the poorer coke) to keep the furnace in good condition. The slag produced required a high temperature for its formation, and a still higher one to make it flow properly — probably some 1350 deg. C. As it flowed freely, and with ample heat to spare, it is probable that it attained a temperature of at least 1450 deg. C. within the furnace.

Metallurgists were inclined to believe that the ability to produce, with moderate fuel consumption, a slag so low in iron and so high in alkaline earths was due largely to the fact that fully one-third of those earths consisted of magnesia — thus multiplying the number of bases in the slag, and presumably increasing its fusibility. The incorrectness of this conclusion will be shown in the succeeding account of analogous work done in later years at the neighboring Rapid City smelter.

Next to the composition of the slags, the most interesting features presented by the experience at the Deadwood smelter are two in number: the production of considerable amounts of metallic iron, which acted more or less as a collector for the gold under certain conditions, and the extraordinary ratio of concentration attained at times.

The subject of the production of metallic iron in smelting pyrite ores, with highly silicious slags, is a most interesting one; and any attempt at discussing it exhaustively would be entirely out of place in this book. I may mention, however, one or two well established points which have a direct bearing upon practical work. In the first place, the addition of more pyrite, which

would be the first impulse of the metallurgist if he wished to check the formation of sows, is usually followed by an increase, instead of a decrease, in their production. This was the case with Dr. Carpenter; it was the case with Samuelson at Kongsberg; it is the case at Mansfeld under slightly different conditions; I reported this fact (without understanding its significance) at the Mount Lincoln smelter, Colorado, in 1872, and I have brief notes of other instances. Without mentioning the various theories that have been offered to explain this curious phenomenon (most of them showing an inadequate knowledge of the conditions under which they form), I will state as my own belief — and this same explanation may, very likely, have been previously suggested — that this production of metallic iron comes mainly from the dissociation of the FeS by prolonged heat. Most blast-furnace mattes contain a certain proportion of metallic iron. Their sulphur content is seldom sufficient to go around, and the metal which is short of its proper quota of sulphur is the iron. The comparatively slow smelting and unusually high temperature of this process subject the small proportion of FeS present to very trying conditions, and the remarkable thing is that not *more* sows are made in ordinary partial pyrite smelting where the temperature happens to be particularly high and the slag slow and refractory. One of the principal reasons that this does not occur more frequently is that ferrous sulphide has the power of holding a considerable proportion of metallic iron in solution, and it is not until its saturation point is reached that the surplus metallic iron begins to separate in the form of slowly growing sows.

Dr. Carpenter says, "The percentage of matte made at Deadwood is very low, seldom exceeding 5 per cent of the weight of the charge, and formerly fell sometimes to 2 or 3 per cent — and here I wish to note a peculiar experience. Before the use of copper ores we could not always control the percentage of matte to be made. Sometimes almost the whole of the iron would go into the slags, and no matte would be made. At such times one would expect that the slags would carry the whole of the gold, but such was not the case. They were not noticeably more foul than at other times. I note one run of 18 hours without matte, while it was common for hours to pass with little or no matte appearing. 'Sow' was doubtless formed; but, if so, it remained in the furnace, and with it the gold.

"It was my theory that this metallic iron helped to clean the slags, and I did not really care to overcome its formation. I believed that if the gold were not recovered in the sows, it would be lost in the slags, and here is, in my opinion, the solution of the question of clean slags in the use of pyrite free from copper as a collector. The Deadwood iron matte was never a pure monosulphide, but a subsulphide carrying metallic iron, some of which could be separated as 'metallics' by grinding and sifting." [1]

The second notable feature to which I referred in discussing the Deadwood matte is the unusual ratio of concentration.

This was intended, originally, to be about 20 into one, and would sometimes remain at this standard for days together, when, suddenly, the ratio would double or treble, reaching occasionally 100 to one, or even higher. This, however, as explained in the above quotation from Carpenter's paper, was more apparent than real, as it simply meant that the iron which, normally, remained in combination with sulphur to form the regular matte, was losing its sulphur (probably largely by direct dissociation) and was gathering in the hearth as metallic iron.

This remarkable fluctuation in the concentrating power of the furnace is one of the characteristic features of the process when an earthy slag with small quantities of iron matte is produced by the partial pyrite method. The addition of copper ores had a most steadying effect upon the production of matte. Cuprous sulphide is not broken up by heat alone to any recognizable degree, and its mutual solubility with ferrous sulphide seems to protect the latter from dissociation.

Having become familiar with the peculiar metallurgical conditions existing in the Black Hills district, we may examine briefly the later work done at the National Smelter, in the same country and under similar conditions in most respects. There are, however, one or two differences which have an important bearing upon the work, and which must be given in some detail.

Perhaps the most interesting of these differences, from a metallurgical standpoint, is that the limestone used as a flux was practically free from magnesia, thus completely demolishing the

[1] Since Dr. Carpenter wrote this, in 1899, I think it has been conclusively established that there is no such compound, in mattes, as a subsulphide of iron, and that the shortage of sulphur so frequently noticed in apparently homogeneous ferrous sulphide mattes is due to dissolved metallic iron.

argument that Carpenter's unusual slags were commercially feasible because his limestone carried so much magnesia.

The composition of the limestone used as a flux, at Rapid City, as given by Fulton & Knutzen, is as follows: SiO_2, 1.94; MgO, 0.68; CaO, 53.61. This is a limestone of unusual purity, and certainly renders a minimum aid toward multiplying the bases in the slag.

The gold ores were mostly free from sulphur and iron, and were highly silicious, as in the preceding case. The pyrite and pyritous fluxes were barren and silicious. The copper ore which was added to improve the collecting power of the matte came from Butte, and was unprofitable, owing to its distant source; its composition is given as follows:

SiO_2	31.62	per cent
Fe	21.47	" "
S	22.85	" "
Cu	24.16	" "
	100.10	" "

Wyoming coke of poor quality (30 per cent ash), costing $4.50 per ton, was mixed in the proportion of two to one with Colorado coke (12 per cent ash) costing $9.50 per ton. The fuel, therefore, was decidedly poor.

The blast-furnaces present no unusual features. One had a tuyere-section of 48 by 144 inches; the other being 36 by 120 inches. The tuyeres had a diameter of five inches, and the average smelting-column a height of about 9 feet. It is considered that this height might be increased four feet with advantage. The blast-pressure at the tuyeres was 16 to 18 ounces.

The blast was pre-warmed rather than preheated, the stove being insufficient for the duty. With an outside temperature of 18 deg. C., the blast at the tuyeres averaged about 55 deg. C.; but even this inadequate heating had a distinctly beneficial effect upon the behavior of the furnace, showing how close to the danger line the smelting was conducted.

The furnaces were semi-mechanically charged by means of a specially designed bottom-dump car which runs directly over the throat of the furnace, the gases being withdrawn below the floor. Fulton and Knutzen state that this method of charging was found eminently satisfactory, and I invite particular attention to the circumstance, because it is a common belief that mechanical

charging will not give satisfactory results in difficult and delicate smelting, especially if there is much fine material present. The case now under discussion possesses both of these characteristics in a marked degree, the process itself being of a peculiarly sensitive and critical nature, while the proportion of fines in the charge is so large that, even with the light blast pressure employed, 10 per cent of flue-dust is produced. Lacking a reverberatory furnace, this material was moistened and re-smelted in the blast-furnace.

The following analyses are given by Fulton and Knutzen as representing typical slags from this smelting:

Description	SiO_2	FeO	CaO	Al_2O_3	ZnO
Blowing-in slag	42.65	16.67	30.7	6.85	—
Typical slag.....	47.5	18.7	28.25	3.5	—
Typical slag	50.2	16.35	28.30	4.2	—
High Al_2O_3 & ZnO	42.86	17.24	26.61	9.39	2.28

In order to economize copper, and to produce a matte high in gold without carrying the first concentration to unpracticable limits, the first matte was generally re-smelted twice. When using a certain proportion of copper ore, which became eventually the regular practice, the matte-fall, figured on the total furnace charge, was between four and five per cent; it contained about 12 per cent copper, which was sufficient to reduce the slag values to about 30 cents per ton in the precious metals, the matte carrying four to five ounces gold and six to seven ounces silver per ton. It was considered that a matte-fall of three per cent, on the weight of the total charge, would give clean slags, provided copper was present as already stated. Before copper ores were used, an iron matte containing seven ounces gold and 10 oz. silver per ton would be accompanied with a slag carrying one to two dollars per ton in precious metals. As Dr. Carpenter had already found, the production of an appreciable amount of metallic iron would correct this evil, but, in this kind of smelting, the degree of oxidation is not sufficiently within the control of the operator to permit of such innovations.

The quantity of copper required to produce a clean slag was the subject of careful study, as there was a financial loss on the copper ore added. It was found that 10 lb. of copper should be present to each ounce of gold (146 copper to 1 gold, by weight), and that more was still better. The recovery of the silver was more difficult than that of the gold, as is universally recognized.

Mattes carrying less than 20 per cent copper were not entirely satisfactory as collectors for the silver.

Eighteen to 20 per cent of (poor) coke was required for these refractory mixtures; yet, in spite of this large proportion of carbonaceous fuel, the desulphurization was some 75 per cent, due largely to the finely divided condition of the sulphides. Thus some 70 to 75 per cent of the sulphur contained in the charge was removed during the first smelting.

The smelting was slow, due to the nature of the slags, and also to the light blast which had to be used to avoid the excessive production of flue-dust. About 105 tons per 24 hours for the small furnace, and 130 tons for the larger, was the average duty, or something like 2.7 tons per sq. ft. of hearth area (26.35 m. t. per sq. m.).

The type of slag produced in later years is still more interesting than that already given, owing to the even lower iron and higher lime which experience had taught to be feasible. Professor Fulton has sent me a record of the daily analyses of the day and night slags for the months of October and November, 1902, from which I have made some approximate averages, omitting a few irregular types.

Subject to the exception mentioned later, the average composition of the slag was approximately:

SiO_2	48.5
FeO	11.6
CaO	33.0
Al_2O_3	5.5
RO bases to balance	1.4
	100.0

In spite of the large proportion of SiO_2, this slag has only about 1.7 times as much O on the acid side as on the base, and would come well within the limits of reasonable slags, were it not for the very low proportion of FeO and correspondingly high CaO.

A slag of this composition would run for 10 to 12 days in a satisfactory manner, when the furnace would begin to show signs of distress. The smelting-zone would creep higher up in the shaft, diminishing the intensity of the focus; the tuyeres would grow dark and the tonnage would diminish. This necessitated an increase in the pyrite flux sufficient to raise the iron content of the slag

materially — perhaps to 16 or 18 per cent. After two or three days of this corrective treatment — during which the normal smelting was in no wise interrupted — the furnace would resume its proper condition, and would be again ready for its regular charge.

In a recent personal letter, Professor Fulton sums up his ideas regarding this form of smelting in a manner so pertinent to the purpose of this book, and so in harmony with my own views, that I give it, in condensed form, as a fitting conclusion to this brief description of partial pyrite smelting as developed in the Black Hills region.

He believes that in smelting ores low in sulphides the main essentials to success in making these slags so high in silica and lime, and so low in iron, are: (*a*) Coke of good physical and thermal quality, which should persist down to the tuyeres, and should approximate 14 per cent of the weight of ore and flux. (*b*) Moderate blast: else, the heat creeps up, and the tuyeres chill. This means also moderate tonnage, seldom exceeding four tons per sq. ft. hearth per 24 hours. (39 m. t. per sq. m.). (*c*) Extreme care in charging and in maintaining regular height of ore-column. Proper distribution of the ore and avoidance of large pieces of ore or limestone. (*d*) Periodical lowering of the silica and lime content of the slag by additional iron whenever the furnace shows signs of distress; *i.e.*, creeping-up of the smelting-zone and chilling of the tuyeres.

There is no doubt, I think, that the preheating of the blast aids this process materially, and sometimes becomes indispensable, and it is also probable that a higher smelting-column — perhaps 12 to 16 feet — would be of distinct advantage after the technical details of the operation had adapted themselves to the new conditions.

This kind of smelting is done from necessity and not from choice, but it is well to know that, if occasion demands, a 48 per cent silica slag with low iron and high lime is not prohibitive.

CHAPTER X

REVERBERATORY FURNACES

THE reverberatory is distinguished from the blast-furnace mainly by the fact that its fuel is burned apart from the ore, and not in direct contact with the latter, as in the blast-furnace. A separate chamber for the combustion of the fuel is not an essential feature of every reverberatory furnace, for where gas or oil is employed, it may be introduced direct into the hearth-space. Most reverberatories, however, employ coal (occasionally wood) as their source of heat, so that it is customary to regard this type of furnace as consisting of two compartments: a large compartment called the hearth, where the ore is smelted; a smaller one termed the fire-box, where the fuel is burned. The ratio between the area of the grate and that of the hearth — modified also by the intensity of the draft — determines the degree of temperature that may be maintained upon the hearth.

It follows, therefore, that reverberatories which are to be used for operations requiring only a moderate temperature, such as roasting (600 to 900 deg. C.), are equipped with a comparatively small grate; while those designed for smelting, where a temperature of 1500 deg. C., and upward, is demanded, are provided with a grate of larger area, and with the means of producing a powerful draft.

In all kinds of reverberatory furnaces a flat, continuous arch is thrown over both hearth and fire-box, while a draft is established by a chimney at the end of the hearth farthest from the fire-box. The draft creates a powerful suction by which the gases supplied by the burning fuel in the fire-box — which acts as a gas-producer — are drawn through the hearth, filling it with flame, and giving up their heat to any object with which they come in contact. In its passage through the hearth, the flame comes in contact with three surfaces: the arch of the roof, the side-walls, and the surface of the hearth itself. The latter is kept covered with a layer of the ore which we desire to smelt,

while the side-walls and arch are necessarily left naked to the contact of the flame and must, in consequence, be constructed of material which shall have a reasonable margin of safety beyond the highest temperature that is to be produced.

A furnace that is to be used for smelting, therefore, requires a widely different construction from one that is to be exposed only to the moderate heat of the roasting process, so that it is more convenient to treat of each class of furnace by itself. The present chapter deals solely with reverberatories for *smelting*.

The simplest means of studying the general features of any apparatus is to determine in advance the precise purpose to which this apparatus is to be applied. Modern practice has demonstrated that the only function which so expensive and highly-organized a structure as the reverberatory smelting furnace can exercise with economy is to produce heat.

Incidentally, two other important results are attained while smelting copper ores in this manner: namely, the removal of some 20 to 25 per cent of the sulphur content of the charge, and an excellent separation of the slag and matte. These latter results, however, should be regarded as beneficent accidents, and quite subordinate to the one main function of the apparatus.

Realizing, then, that the production of heat is the one fundamental duty of the furnace, and that the amount of ore smelted depends absolutely upon this heat; knowing also that the maintenance of the furnace and the wages of its attendants constitute a fixed, average daily expense which remains practically the same whether much or little useful smelting is accomplished: it follows that the profits of the furnace are proportionate to the rapidity with which ore is smelted; further, that the rapidity with which ore is smelted depends upon the degree of temperature maintained upon the hearth; and, lastly, that the heat upon the hearth depends upon the rapidity with which coal is burned upon the grate.[1]

In burning coal, or other fuel, in a reverberatory furnace it is not enough to take into account the number of heat-units produced; we must also take care that a certain *intensity* of heat is attained. For instance, an average ore may melt satisfactorily at a temperature of 1150 deg. C. We might burn sufficient coal

[1] Discussions and demonstrations of the truth of these propositions may be found in the author's "Principles of Copper Smelting," Chapter VII.

on the grate of our furnace to heat a vast quantity of this ore up to a temperature of say 1050 deg.; yet no useful result would have been accomplished. Our standard of intensity is too low.

In the older reverberatories, this standard of intensity was so planned that, under favorable conditions and with great expenditure of fuel, we could finally reach our temperature of 1150 deg., — and even get a little beyond it — with the result that we could eventually smelt our ore; but it took an exceedingly long time, and any slight disturbing circumstances prevented the complete attainment of the required temperature, or delayed it beyond all reason. It must be noted that it is only the last fifty or one hundred degrees of heat that does the actual melting. All the rest of the heat is employed in warming the ore up to the point where fusion commences, and where profits begin to become apparent. Consequently, our best attention should be focused upon that last few hundred degrees of heat which is the sole source of dividends.

The modern reverberatory has raised the average standard of quick intensity by several hundred degrees, and now maintains a temperature of 1500 to 1575 deg. C. at the fire-bridge. This gives us some 400 deg. of leeway on our 1150 deg. ore; so that instead of the actual fusion of the ore being the terminus of our progressive heating, it is now only a way-station at which no stop is made; and the superheating of the molten ore — as well as of the inner walls and arch — continues on to an extent that would soon become dangerous were it not for the frequent addition of large quantities of fresh ore to absorb the heat from the superheated brickwork which encloses it, and from the superheated slag and matte on which it floats. Indeed, to those whose active work in reverberatory smelting lay mostly within the older lines, it seems almost that, whereas with the former furnaces we were always adding coal in a vain endeavor to get our furnace hot enough, the present smelters are always adding ore to keep the furnace from melting down.[1]

[1] The rapidity with which heat is absorbed by one substance from another increases enormously as the difference in temperature between the two substances becomes greater. While this law cannot yet be expressed in exact figures, careful experiments show, for instance, that if we have a fresh charge of ore with a melting-point of 1150 deg. C., the time required to fuse it with a flame having a temperature of 1400 deg. C. may be two hours; but with a

This splendid result has been accomplished, in the main, by two great and progressive changes in the management of the furnace: (*a*) burning coal more rapidly; (*b*) conserving the temperature within the furnace by various measures which prevent ingress of cold air and egress of heat.

The following table illustrates some of the points to which I have just referred, and shows especially the remarkable increase in economy resulting from the more rapid combustion of coal in modern practice:

Furnace	Length of Grate, Ft.	Breadth of Grate, Ft.	Area of Grate, Sq. Ft.	Coal per 24 Hours, lb.	Coal per Minute per Sq. Ft. Grate-area, lb.	Ratio of Area of Chimney to Grate	Ore Smelted per 24 Hours, lb.	Lb. Ore Smelted by 1 lb. Coal
Argo, Col., 1887	5.5	4.50	24.75	18.000	0.505	1: 2.75	48.000	2.67: 1
Argo, Col., 1891	6.0	4.75	28.50	20.000	0.487	1: 3.17	56.000	2.80: 1
Argo, Col., 1894	6.5	5.00	32.50	27.000	0.577	1: 2.03	100.000	3.70: 1
Montana . . 1903	10.0	5.50	55.00	72.000	0.909	1: 1.83	224.000	3.10: 1
New Anaconda	16.0	7.00	112.00	114.000	0.707	[1]	550.000	4.82: 1

Assuming still that our first object is to burn coal as rapidly as possible, the most important lesson that we can learn from this table is that the weight of coal burned in a given time increases in inverse proportion to the ratio of area of grate to chimney; or, put in plainer language, the larger the area of the chimney in

temperature of 1550 deg. C., the time required will not exceed one hour. This statement is only an approximation of the truth, but is believed to be well within bounds. No stronger argument can be adduced in favor of employing high temperatures for smelting copper ores in the reverberatory furnace.

[1] In the case of the Anaconda furnace, it is impossible to state the ratio between grate area and stack area, as the gases discharge into an extensive system of flues, leading to a central chimney 30 ft. in diameter, and 300 ft. in height, which is common to the entire plant. It may be said, however, that the draft is much stronger than in any of the other furnaces tabulated, corresponding to about one and one-half inches of water, in the downtake of the reverberatories. The waste gases from one of these large furnaces furnish about 600 horse-power in generating steam, and some 13 tons of coke and coal fines is recovered daily from the ashes by jigging. This is briquetted with flue-dust and fine ore.

proportion to the area of the grate, the more rapid will be the combustion. This is shown clearly in the following arrangements of figures from the table just given:

Ratio between Area of Chimney and Grate		Lb. Coal Burned per Minute per Sq. Ft. of Grate
1 : 3.17	=	0.487
1 : 2.75	=	0.505
1 : 2.03	=	0.577
1 : 1.83	=	0.909

We see, then, that when the grate had an area 3.17 times as large as the chimney, the coal burned per minute per square foot of grate surface was only 0.487 lb. Going to the other extreme of the table, we find that when the area of the chimney was so increased that it was considerably more than half as large as that of the grate — 1: 1.83 — the combustion of coal rose to 0.909 lb. per minute per square foot grate area.

Modern practice, therefore, emphasizes above all things the importance of burning coal rapidly upon the grate of the reverberatory furnace, while the laws of combustion point out the one essential condition for the accomplishment of this purpose: namely, a sufficiently strong draft. This is, perhaps, the most fundamental requirement for economical reverberatory smelting.

The second requirement for attaining capacity and economy in this process is that the heat already attained within the furnace shall be conserved so far as possible.

This apparently simple demand covers a multitude of details, the more important of which must be understood before the study of the construction of the furnace can be pursued to advantage.

Assuming that a large and constant evolution of heat is occurring within a reverberatory smelting furnace, and that the aim of the metallurgist is to conserve this heat (for smelting his ore) so far as possible, he should first establish clearly in his own mind what is becoming of this vast quantity of heat which is being generated. It certainly does not accumulate indefinitely within the furnace and must, therefore, find certain avenues of escape. Some of these avenues — such as absorption of heat by the ore-charge — are legitimate, as they lead directly to the object which the smelter is trying to attain. Others are unavoidable. Still others cannot be closed entirely, but may be greatly diminished.

The more important avenues of escape for the heat generated in the reverberatory furnace are:

1. Heat in the escaping gases.
2. Heat in the molten products.
3. Heat lost by radiation from the furnace.
4. Heat lost by the admission of cold air into the furnace.

1. *Heat in the escaping gases.*—The temperature of the escaping gases must be sufficiently high so that they may not cool the front end of the furnace to a point where the slag is not sufficiently liquid to flow readily. That is to say, there is no advantage in building the hearth of the furnace so long that the front section of it acts as a refrigerator instead of as a heater. The slag is superheated in the intensely hot portion of the furnace near the fire-bridge. Much of this superheat is recovered by being given up again to the fresh ore-charge that is dropped upon it; and a certain amount of the excess heat is left in it to enable it to separate thoroughly from the matte as it is pushed gradually toward the remote skimming-door by fresh accessions of ore at the bridge-end of the furnace. Sufficient excess heat should still be retained so that it may maintain its complete liquidity until it has flowed out of the furnace and reached its final destination in pots or granulating-gutters.

Under ordinary conditions, this means that the temperature of the escaping gases at the flue-end of the furnace may not be much below 1150 deg. C. An immense amount of heat is thus unavoidably withdrawn from the furnace, and, although valueless for further direct application in the process, may be utilized most advantageously in the production of steam for power. This utilization of the escaping gases for the production of steam has become an almost essential feature in modern reverberatory practice.

2. *Heat lost in molten products.*—This is another unavoidable avenue whereby a vast amount of heat is removed from the furnace. The amount of energy represented by the heat stored up in the two hundred tons, or more, of slag and matte produced daily by a large reverberatory furnace is astonishing. A portion of this heat, although lost to the furnace, is utilized in succeeding operations. Thus the matte is tapped, or poured, direct into the converters, saving the considerable amount of fuel, labor, and apparatus that would be required for its remelting. The slag,

however, which represents much the more important loss, is seldom utilized as a source of heat, although late inventions offer some encouragement in this direction.

3. *Heat lost by radiation from the furnace.* — A certain amount of loss in this direction is unavoidable, being, to some extent, essential to the safety of the furnace. Fire-brick are such poor conductors of heat that, when laid in the shape of a tolerably thick wall, there is not much more heat lost from the outer surface than is essential to keep the inner surface from suffering from the high temperature of the interior of the furnace.

Where the radiation seems excessive, an enclosing wall of red brick will correct it. In this respect, therefore, there has not been much room for improvement upon the older methods of construction.

There is, however, one source of loss by radiation which has been materially diminished of late years. This is the open cooling-space, or vault, which was a constant feature of the older furnaces. Improved construction and management have eliminated the necessity of cooling the very portion of the furnace which was always too cool anyway for rapid fusion, and have thus diminished materially the loss of heat by radiation, besides increasing the rapidity and effectiveness of fusion to a remarkable extent.

4. *Heat lost by admission of cold air.* — It is in this section of losses that modern practice has achieved its greatest improvements. These advances can be appreciated at their full value only by those personally familiar with former methods of operation. They are so numerous and so far-reaching that[1] their consideration would involve a detailed study of the entire practice now adopted at modern smelting establishments, as well as of the improvements in construction which render it possible to carry out the modern practice. These subjects will be taken up later; at this point I will merely say that cold air is now excluded from the furnace by keeping the working-doors shut, and the grate always well filled with coal in a condition of active combustion; and that we are enabled to keep the working-doors shut and the fire active because we have learned that the operations of charging, firing, skimming, and tapping may be executed almost without opening the furnace-doors, or materially cooling

[1] See "Principles of Copper Smelting," pp. 180, 181, for a detailed comparison of old and modern practice in reverberatory smelting.

the hearth; while fettling may be omitted altogether for a fortnight or more.[1]

The construction and the operation of a furnace are subjects so intimately connected that, until it comes to the actual details of building, it will be convenient to consider them together in these introductory remarks.

According to older methods, the routine of work at the reverberatory consisted of a series of rather distinct operations, separated by considerable intervals of time, and involving much opening of working-doors and consequent admission of cold air into the hearth. These intervals of time — employed for charging, leveling, stirring, skimming, tapping, fettling, etc. — made a serious inroad upon the actual smelting-time of the furnace, but were trifling as compared with the delay caused by the consequent cooling of the hearth and charge.

When the degree of heat required for the fusion of the ore corresponds pretty closely to the maximum temperature that can be attained only by long, uninterrupted firing, and by a conjunction of favorable conditions, it is fatal to allow unnecessary cooling; for while the mere opening of the working-doors for a few moments will cause a drop of 200 deg., or more, in the temperature of the hearth, it may take the better part of an hour to recover this heat.

It is obvious that much of the time and fuel expended in the older practice was consumed in recovering the heat lost by opening the working-doors; also, that the opening of the working-doors was a frequent necessity for the accomplishment of four principal duties:

1. Spreading and leveling the charge.
2. Stirring the partly-fused charge to bring up the half-melted masses sticking to the bottom.
3. Skimming the slag. (Often accompanied by subsequent deliberate cooling in order to chill the last vestiges of slag sufficiently to get a clean tap of matte.)
4. Fettling the hearth.

[1] The skimming-door, which has to be opened for the removal of slag four or five times in 24 hours, admits but little cold air into the furnace, as it is situated directly beneath the flue-opening, so that the outside air which rushes in is sucked directly up the flue — checking the draft slightly, but otherwise having but little cooling effect upon the hearth.

A consideration of these four subjects will include the technical management of a reverberatory furnace according to present methods, and will also indicate the changes in its construction which have been found necessary to permit this improved management.

1. *Spreading and leveling the charge.* — This tedious and laborious duty has been almost, or entirely, eliminated in modern practice by keeping the hearth constantly filled with exceedingly hot matte and slag. The new charge, dropping upon this molten lake, spreads itself with little or no attention.

In the majority of instances this automatic spreading action is aided materially by the fact that a considerable proportion of the new charge consists of hot ore from the roasting furnaces, which flattens and spreads itself in a most thorough and striking manner, and does not demand even the momentary opening of a working-door.

We may say, then, that the former serious cooling-effect due to the operation of charging and leveling has now been eliminated by keeping the hearth well filled with the normal molten products, at the same time using the roasted portion of the charge in a heated condition.

The practical smelter of the preceding generation did not fail to recognize, to a considerable extent, the advantages gained by keeping his hearth full of matte and never tapping the furnace dry excepting when required by repairs. He was, however, always in fear of injuring his bottom, or of experiencing outbursts of matte through the side-walls unless he cooled his furnace occasionally; and the constantly-recurring operations of mending and fettling demanded an empty hearth. The manner in which these difficulties have been solved is the key to modern practice.

2. *Stirring the partly-fused charge to bring up the half-melted masses sticking to the bottom.* — This harassing and unworkmanlike duty was formerly a part of the routine work of nearly every reverberatory furnace, and lowered the temperature of the interior in a manner that was most painful to every superintendent. If omitted, it was only delaying the evil moment, as thick hearth-accretions formed, foul slag resulted and, eventually, normal smelting had to cease until the crusts were melted out by long-continued firing. Assuming a properly-fluxed charge, the common causes of these accretions were:

(*a*) Want of heat — this was almost invariably the fundamental reason for the half-fused, sticky masses which encumbered the hearth.

In the first place, the vault-cooled hearth — chilled still further by open doors and cold ore — was blanketed by the fresh charge, and could only become warmed by the small proportion of heat that penetrated slowly through the latter. Again, the normal temperature of the furnace was so low that, after the more easily melted constituents had come together and liquated away from the less fusible portion, there was not heat enough to continue the operation at an increasing temperature, and force the already melted portion to dissolve and assimilate the refractory residue. This latter adhered persistently to the bottom and, unless broken and detached by the rabble, and thus brought to the surface, was likely to require long firing to remove it.

This trouble is now almost completely obviated by the melted matte upon which the new charge floats, and which yields up some of its superheat to the latter, so that the new charge is not only prevented from sticking to the bottom, but is literally between two fires — being heated from below by the matte upon which it floats, and from above by the flame.

(*b*) Imperfect mixing of the constituents of the charge. — Smelters are recognizing more and more the important part played by *propinquity* in the various smelting operations. A fragment of quartz and a fragment of iron ore which touch each other will form a fluid slag more quickly and at a lower temperature than if separated. A better appreciation of this fact leads to increased care in the mixing of reverberatory charges, and it is now common to feed even fluxes (crushed limestone) into the roasting furnace along with the sulphide ores in order to obtain a thorough mixture, as well as to preheat the cold material.

We may say, then, that the adherence of half-fused masses to the hearth is now obviated by the permanent pool of matte already referred to, and also by a thorough preliminary mixing of the component parts of the charge.

3. *Skimming the slag.* — In former days, if we attempted to smelt three or four successive charges without skimming the slag already formed, we always experienced a series of troubles and delays that promptly discouraged the innovator.

One main difficulty was that the cold charge absorbed at

once the heat from the melted slag, and soon reduced the temperature so far that the latter chilled into a solid crust which included the underlying, comparatively small body of matte. This chilled crust, blanketed on its upper surface by the fresh charge, and receiving little aid from the air-cooled hearth upon which it rested, was pretty thoroughly protected from heat — and behaved accordingly. My own experience taught me that no time was gained by "doubling" charges in this manner and that, with the type of reverberatory then in use, the best thing we could do with a charge that was once thoroughly melted was to get it out of the furnace as soon as possible.[1]

When modern skill had demonstrated that we could, with safety, employ a deep hearth of great capacity, and that smelting upon it could be done for weeks together without the necessity of emptying it for repairs, the whole manner of looking at the smelting operation underwent a radical and remarkable change.

The hearth is now regarded as a great permanent reservoir of melted material to be kept always full. This melted material consists of two substances having different specific gravities and, consequently, arranges itself into two distinct layers, each of which is maintained constantly at somewhere near a fixed level. We may even go so far as to regard this molten lake as an integral part of the furnace, as much as is the sand of the hearth or the fire-brick of the arch. We may say, for instance, that the reverberatory hearth — beginning at the bottom — consists of 28 inches of agglomerated sand, six inches of matte, and about eight inches of slag; the latter substances having a slight periodical variation.

As soon as the last preceding charge of ore has flattened and softened down, and become assimilated, a fresh charge is dropped upon the hottest portion of the hearth (near the fire-bridge), and so the process goes on continuously. Frequent firing is of course required to maintain the normal temperature of the interior, and — incidentally — when the hearth threatens to overflow, or when the converters demand a few tons of matte, enough is withdrawn to satisfy the situation. It is not until the old idea

[1] These remarks apply only to the slag. It was nearly always found advantageous to let the matte accumulate for several charges (if the hearth would hold it safely); but the repairs required by the hearth prevented us from ever realizing the full advantages of this practice.

of periodicity is given up, and the new idea of continuity is adopted, that the mind can grasp satisfactorily the present practice of reverberatory smelting.

The withdrawal of a few tons of matte is, naturally, a simple matter, as the tap-hole is situated at the deepest part of the hearth, and is closed with ease, and without affecting the running of the furnace, whenever the desired quantity of matte has escaped.

The withdrawal of the slag, with a minimum effect upon the continuity of the operation, is accomplished by maintaining the normal slag-level at some distance above the skimming-overflow by means of a rude dam of sand. Upon the gradual removal of the latter, at intervals of several hours, the slag flows out rapidly and quietly, until the proper low-water mark is reached in the hearth.

It will be seen, therefore, that the cooling effect due to the operations connected with the skimming and tapping of the furnace has been almost entirely eliminated by maintaining a deep hearth full of the molten products, and allowing the latter to flow off by gravity when too abundant, without opening working-doors, or without interrupting the normal, continuous operation of smelting.[1]

4. *Fettling the hearth.*—The repairing of the hearth and its walls was formerly an imperative and constantly-recurring duty. The principal injury which the hearth suffered from the normal smelting operation was of a two-fold nature: (*a*) destruction of the hearth proper (sand-bottom); (*b*) the undercutting of the lining walls at their junction with the sand-hearth.

(*a*) Destruction of the hearth proper.—In the ordinary reverberatory furnace, smelting is conducted upon a hearth, or bottom, consisting mainly of silica in the form of sand, or of crushed rock. When a fresh charge of roasted ore is spread upon this naked surface of heated silica, the iron oxides of the ore corrode it quickly, forming fusible silicates and eating holes into its less compact portions. This not only destroys the desired smoothness of surface, but also gives the matte an opportunity to burrow deeply into its structure and, eventually, to float up great slabs, or sections, of the agglomerated sand.

This trouble was met by frequently throwing in sand or silicious ore upon the damaged portions of the hearth, causing delay and loss of heat, and tending toward the formation of

[1] See note on p. 318.

sticky, silicious crusts which delayed the completion of succeeding charges. Still more serious delay and expense resulted whenever any radical patching of the bottom became necessary.

(*b*) The undercutting of the lining-walls near their junction with the sand-hearth. — A line of junction is always a weak place, and is doubly so at the junction of two substances exposed to intense heat, and having a different factor of expansion. In spite of careful protection with clay and sand, the fluxing action of the ferruginous slag would rapidly cut a deep groove at the horizon extending all around the edge of the hearth. If this was not repaired promptly, there was danger of undercutting the lining-walls of the hearth until they fell bodily inwards.

The matte also was inclined to use this groove as a means of bursting through the side-walls, or of working down between the walls and the hearth and floating up the latter.

These serious accidents seldom occurred to experienced furnace-men; but their absence was ensured only by constant fettling and repairing, and at a cost of delay, labor, and material that formed a heavy handicap to the process.

The chief agent in their elimination has been the practice of keeping the hearth full of matte so that it is completely protected from contact with the bases of the charge; further, the constructing and maintaining of a broad, heavy border of silicious material — usually silicious ores — which protects the contact-horizon completely from any attack of the slag. In addition to these measures, unusual care is taken to produce a slag of uniform and non-corrosive character, and to mix the charge so carefully that isolated patches of abnormally basic slag shall not occur.

All of these points will receive consideration in subsequent sections.

Having enumerated and considered the principal duties which the smelter expects of the reverberatory furnace, we may next examine the present means at our disposition for the fulfilment of these demands. These fall naturally into two groups: the first being the construction of the furnace; the second, the management of the process.[1]

[1] The principles of construction and management remain practically the same whether the furnace is of full modern size or whether it is smaller. As it is impracticable to offer plans and instructions for various sizes, it will

The first step to be taken in the construction of a new furnace is to determine its size. Assuming that the supply of ore is ample and that fuel and building material are available, experience shows that size and economy advance hand in hand until certain limits are reached. These limits must now be considered.

Regarding the reverberatory furnace roughly as a rectangle, its two most obvious dimensions are its length and its width. Experience teaches us that its width should be as great as is compatible with economical manipulation and, subordinately, with the maintenance of the broad, flat arch which spans it from side to side. The latter point is subordinate because the limit of economy of manipulation is reached before the arch becomes broad enough to cause serious difficulty. This manipulation consists mainly in operations connected with the periodical repairing of the hearth, and which demand that the fettling-material be thrown through the working-doors, across the hearth, in order to reach the opposite side.

The regular width of the present full-sized reverberatories is about 19 ft. (5.8 m.) inside measurement. This statement, however, does not mean that the hearth possesses any such effective width as the above. A newly-fettled 19 ft. hearth has seldom an actual smelting-width of more than 16 ft. (4.9 m.). As erosion progresses and greater area is obtained, the tonnage of the furnace increases; and this circumstance suggests that — with other dimensions undisturbed — a moderate increase in width should be advantageous, provided that the iron-work be strengthened accordingly. I do not see why a 24 ft. (7.3 m.) hearth should be at all excessive if the fuel is of good quality. The difficulty of fettling so wide a hearth may probably be surmounted by measures already tried successfully at several smelters. These measures will be described later.

The type of furnace, however, to which the present remarks especially appertain will be the regular Mathewson 19 ft. pattern, in use at Anaconda, Garfield, Cananea, Humboldt, and elsewhere.

Having thus determined that the width of the furnace shall be as great as is compatible with comfortable manipulation, we may decide that its length shall be as great as is compatible with the economical heating of so long a structure; so that this dimen-

be more serviceable to describe the larger type, and indicate from time to time any important differences which distinguish the smaller furnaces.

sion is determined mainly by the character of the fuel. The gases, as they leave the front end of the furnace, must be sufficiently hot to maintain the slag in its proper liquid condition, and yet should not be much beyond this temperature. Yet this high temperature at the front end of the hearth must be natural; that is to say, it must result from the normal burning of the fuel gases at the active portion of the hearth (near the bridge), and not from any artificial restraint of combustion at that important point with a view of maintaining a greater heat at the far end.

The determination of this question of length is one of the most important and difficult points in any new enterprise, and it is better to err on the safe side and make the furnace too short, rather than to construct a hearth so long that it is a constant struggle and delay to keep it hot enough at the skim-door for comfortable work. This subject is intimately connected with the question of fuel, and will be considered under that head.

The Anaconda furnaces have grown by successive stages from 20 ft. to their present length of about 115 ft. (35.1 m.) from bridge to skim-door. This exceptional length is rendered possible by the strong draft and the excellent quality of their highly gaseous coal, and experiments show that no advantage is gained by increasing this dimension. While other smelters have erected furnaces of 98 and 100 ft. (30 and 30.5 m.) in length, I believe that, as a rule, the ordinary, fairly good reverberatory coal of the western United States is better suited to a hearth length not exceeding 90 ft. (27.5 m.).

The size of the fire-box and grate must be sufficient to burn coal with the rapidity needful to maintain with ease the high temperature required for economical work, and also to provide an ample margin for choking with ash and clinkers. This dimension, also, must depend largely upon the physical and chemical qualities of the fuel. It is not enough to know the percentage of fixed carbon, volatile constituents, and ash which the coal contains. Its freedom of igniting and burning, its tendency to decrepitate or to crumble into fines, the fluxing quality of its ash, the tendency of the latter to melt into clinker, and the quality of this clinker (whether brittle or tough) are all factors which have a bearing upon the needful grate area. With good coal, a proportion of grate to hearth area of 1 to 15 might be reasonable. This would have to be considerably increased for poorer fuel.

The accompanying table shows certain interesting features in the evolution of the reverberatory smelting furnace from the small Swansea type of 1878 to the furnaces now in use.

TABLE SHOWING DEVELOPMENT OF THE AMERICAN REVERBERATORIES FROM 1878 TO 1910

Year	Dimensions in Feet			Area in Square Feet			Ratio Between		
	Stack	Fire-box	Hearth	Stack	Fire-box	Hearth	Stack	Fire-box	Hearth
1878	2.75	4.5 × 5	9.66× 15	7.56	22.5	105	1	2.98	13.9
1882	2.75	4.5 × 5	10.33× 17.83	7.56	22.5	139	1	2.98	18.4
1887	3	4.5 × 5.5	12.67× 21.17	9	24.75	198	1	2.75	22
1891	3	4.75× 6	14.17× 24.33	9	28.5	265	1	3.17	29.5
1893	3.5	5 × 6.5	16 × 30	12.25	32.5	402	1	2.65	32.8
1894	4	5 × 6.5	16 × 35	16	32.5	481	1	2.03	30
1903	5.5	5.5 ×10	20 × 50	30	55	840	1	1.83	28
1910	....	8 ×16	19 ×115	.....	128	1967	....	....	.....

Year	Ratio Between		Per Cent Hearth Area Lost in Corners	Ore Smelted per 24 Hours—Tons		Coal per 24 Hours			Tons Ore Smelted per Ton Coal	Pounds Ore Smelted per Sq. Ft. Hearth per 24 Hours
	Fire-Box	Hearth		Cold Ore	Hot Ore	Tons	% Coal for Cold Ore	% Coal for Hot Ore		
1878 ..	1	4.67	28	12	.....	5	42	.....	2.4	229
1882 ..	1	6.18	24.5	17	.....	7	41	.....	2.43	245
1887 ..	1	8	26	24	.....	9	37.5	.....	2.67	242
1891 ..	1	9.3	23	28	.....	10	36	.....	2.8	211
1893 ..	1	12.37	16	35	43	13	37	30	2.7-3.3	174-213
1894 ..	1	14.8	14	.....	50	13.5	.....	27	3.7	208
1903 ..	1	15.3	16	42	70	36	.....	.....	3.1	267
1910 ..	1	15.4	10	.....	275	59	.....	.....	4.66	280

As no cooling-vault below the hearth is desired, the foundation of the reverberatory furnace is simple. Its essential feature is a massive, rectangular slab of concrete — or other suitable material — resting upon firm and uniform ground. Deep excavation is seldom necessary, and the foundation block should be two feet or more in thickness, and may consist of solid concrete, of brickwork or rockwork covered with a layer of concrete, or of slag. The Anaconda furnaces are built upon a monolithic block of slag poured into the excavation while liquid.

The extent of this foundation block, as well as the details of construction, is shown in the accompanying plans of one of the large Anaconda furnaces.[1]

The foundation block is then covered to a depth of some 12 inches with old fire-brick, carefully laid. This layer (sometimes omitted) completes the foundation of the main portion of the furnace, and now reaches the floor-level, or a few inches higher. The entire hearth, side-walls, front-wall, and bridge rest upon this structure.[2]

The side-walls and the front end-wall are 26 in. (0.66 m.) thick, eight in. (0.203 m.) of this being a casing of red brick. These walls, after reaching a height of 12 in. (0.305 m.), increase in thickness toward the interior by a series of overhanging steps, which begin to recede again shortly above the surface of the sand-hearth. This construction, as well as the limits of the red brick casing, is shown on the plans.

The roof or arch is of silica brick on end. Brick 20 in. (0.508 m.) in length are used for the portion of the arch extending over the fire-box, and for a distance of 30 ft. from the rear end-wall. The remainder of the arch[3] is built of 15 in. (0.381 m.) silica brick. The entire arch is independent of the side-walls, taking its bearing against horizontal I-beams which transfer the pressure to the buckstays. Expansion joints are provided in both side-walls and roof, as explained in the table of dimensions.[4] Various

[1] These plans are reproduced from those published by C. Offerhaus in the *Engineering and Mining Journal* of June 15, 1908.

[2] While most of the following directions may vary within reasonable limits, it will be more convenient for descriptive purposes to make them conform to the accompanying illustrations.

Unless otherwise stated, all refractory brick used in construction are the regular Anaconda silica brick made near the smelter. They are reasonably cheap and exceedingly good, and are, consequently, used more freely there than would be the case in most places. Cheaper fire-brick could be substituted in all parts of the furnace that are not exposed to great heat. The Anaconda silica brick consist of pure white quartz ground to pass a $\frac{3}{16}$ in. screen (0.0047 m.) and pugged thoroughly with two per cent of slaked lime. This mixture is pressed into brick, steam-dried, re-pressed by hand, and burned in kilns at a high temperature. It is made into various shapes to avoid cutting.

[3] This improvement is not shown upon the plan.

[4] Experiments show that both silica-brick and ordinary fire-brick, at a temperature of 1150 deg. C., expand about one per cent in all three directions.

openings are left in the arch for the coal- and ore-hoppers, as well as for the admission of air to burn the fuel-gases. The latter openings, about 20 in number, are 3 by 3 in. (0.076 m.) in size, and are spaced across the roof above the bridge.

At about 30 ft. (9.15 m.) from the front end the hearth begins to taper gradually, until it narrows to seven ft. (2.135 m.), the rectangular outside of the structure being preserved by filling in with red brick.

The flue opening has a normal size of 38 by 60 in. (0.965 × 1.524 m.), and an average height of 45 in. (1.143 m.). Its capacity is modified by varying the height of its side-walls. Warm weather demands increased flue area.

The escaping gases are delivered into a spacious brick chamber in which are situated two water-tube boilers, which absorb sufficient heat to reduce the temperature of the gases from about 980 deg. to 350 deg. C., and generate constantly 600 H.P. The tubes are not attacked by the gases, but require a monthly internal scraping and a daily external blowing to remove As_2O_3. A by-pass flue makes it possible to convey the gases direct to the main reverberatory flue, when they are not being used to heat the boilers.

The fire-bridge has three transverse expansion joints, and presents no other unusual features. Perhaps its most interesting point is its small elevation above the grate, being only 27 in. (0.686 m.). It need scarcely be said that this measurement represents the most careful study and experiment extending over long periods of time, and that it applies solely to the particular coal used at the Washoe smelter. I believe, however, that modern reverberatory practice tends to a comparatively shallow layer of coal, and to the abolition of the gas-producer form of fire-box.

The brickwork of the fire-box is shown plainly in the plans, and calls for no especial consideration. There being abundant water at Anaconda, a considerable volume of it is used for carrying away the ashes, clinkers, and fine coal which drop from the grate. This saves labor, keeps the ash-pit cool, and delivers the mixed ashes and half-coked fragments of coal to jigs situated on a lower level, which recover the coal at a trifling cost.

The grate is very simple, consisting of square two-inch (0.051 m.) bars, spaced three inches (0.076 m.) apart.

The ironing of the modern reverberatory must be extraordi-

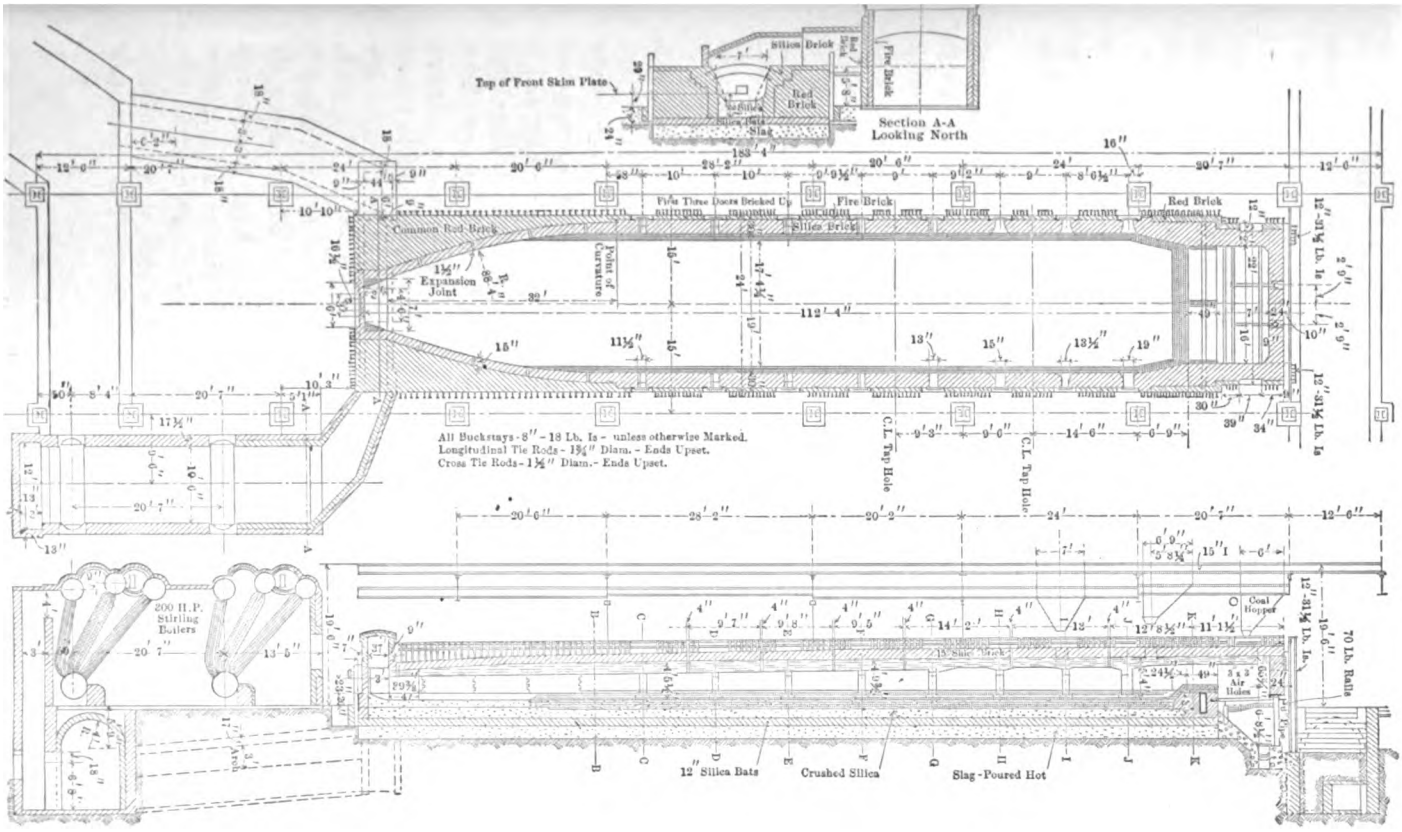


FIG. 69. — Washoe reverberatory — sectional plan and elevation

narily strong to withstand the pressure caused by the excessive temperature, as well as the powerful thrust exerted by the broad and heavy arch. At Anaconda, about 360 eight-inch (0.203 m.) I-beams are used as buckstays for one of the long furnaces, and the upper ends of these are tied by two-inch rods (0.051 m.). No tie-rods are used for the lower ends of the buckstays, as they would suffer from the heat unless placed unreasonably far below the hearth. Instead of tie-rods, the buckstays are held firmly in place by heavy blocks of slag, three feet or more in thickness, and extending — below the floor-level — between adjacent furnaces. These blocks are formed by pouring liquid slag into excavations, or by constructing massive slabs of concrete.

As the furnace does not begin its regular duties until the sand-bottom is in place and beyond danger of injury, it is proper to include the latter among the details of construction. The main office of the bottom is to furnish a support for the great quantity of slag and matte which constantly fills the hearth; therefore the chief qualities demanded of it are that it shall not be injured by heat or by the liquid material resting upon it, and that it shall be cheap. These demands are fulfilled satisfactorily by silica in a condition of tolerable purity — such as quartz sand, crushed quartzite, crushed sandstone, and the like.

For the high temperatures now employed in smelting, it is essential that a durable hearth material should contain a high percentage of silica — probably not below 92 per cent — and better, 95 per cent or more. The so-called Dillon quartz used at the Washoe works contains about 95 per cent of silica, and is a very satisfactory material for the purpose. It is crushed to $\frac{1}{8}$ in. (0.003 m.) and is charged cold into the cold furnace, being tamped firmly as it is introduced. This sand-bottom is about 26 in. thick, and should slope slightly from every direction toward the tap-hole, the rate of inclination being one inch in twenty feet.

The "melting-in" of such a bottom consists in heating it gradually to the extreme limit of safety, by which means the entire mass is consolidated, while the upper layer undergoes a certain amount of agglomeration. After this is accomplished the next step is to preserve intact this superficial crust until the upper sand layers have become thoroughly soaked and cemented with matte, and thus rendered safe and solid. The means used

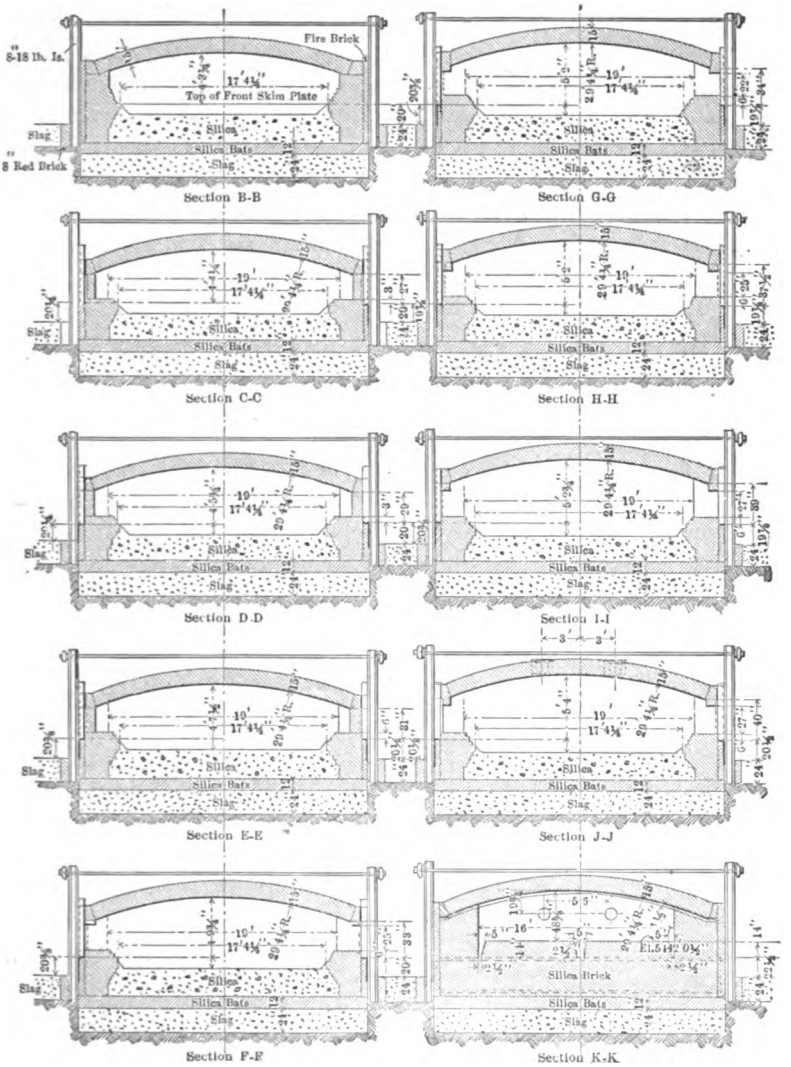


FIG. 70. — Washoe reverberatory — cross-sections

to accomplish this end vary slightly according to local experience and material, but the Anaconda practice will serve to indicate the general plan which has been found suitable for preparing a solid and permanent bottom.

The furnace being dry, but unheated, a moderate fire is started, and increased gradually and steadily for four days, or until a smelting temperature is attained. During this warming-up period, the tie-rods are constantly loosened to correspond with the gradual expansion of the structure, and every care is taken that the ironwork of the furnace shall maintain a steady, equable pressure upon the walls, without permitting any distortion. The roof is allowed to rise about two inches.

At the close of this period, the true "smelting-in" of the sand-bottom begins, and requires about 60 hours of hard and continuous firing. By this time, the silica-brick of the interior are melting perceptibly on their surface, and the upper layer of the sand-bottom is fritted and consolidated to a considerable extent. This plastic crust is now hardened by slackening the fire until the interior shows only a cherry-red heat, and the entire surface is covered some three inches (0.076 m.) deep with silicious, partly-oxidized material, flue-dust from the roasters being generally used at this smelter. This is fused quickly upon the hearth and the regular charges of roasted sulphides are then begun, taking especial care that the matte does not become so heated as to "boil," and thus endanger the protecting layer. For some two weeks this careful practice is maintained, a fresh charge of ore being always dropped into the furnace before the preceding one is thoroughly hot and liquid. At the end of the fortnight the bottom is well soaked and cemented with matte, and by the third week the furnace is up to its normal duty. Furnaces in constant operation for four years (with a duty of about 350,000 tons (317.450 m. t.) of ore, show that the bottom has worn down about two inches (0.051 m.), and it is considered that the normal life of such a bottom should be at least 10 years.

At the first fettling, and as soon as any erosion of the side-walls and bridge begins to show, the brickwork of these walls is built out slightly over the edge of the bottom so as to form a triangular mass which protects the line of junction. When this is well covered with silicious ore, it becomes as solid as the bottom

itself and, except at the bridge-wall, requires no renewal for months at a time.

It is instructive to learn what quality of sand has proved unsatisfactory in large reverberatory ore-furnaces. Mr. L. D. Ricketts reports[1] that the first Cananea (Mexico) reverberatory was started on a bottom composed of sands separated from the tailings of the concentrator, and containing SiO_2, 66.4 per cent; Al_2O_3, 13.1; Cu, 0.8; Fe, 2.6; S, 1.9. This material was notably unsatisfactory, being slagged and floated up almost immediately. A crushed quartzite containing 90 per cent SiO_2 and 2 to 3 per cent Al_2O_3 was tried next, but also was found unsuitable. Finally, a quartzite containing 95 per cent SiO_2 was brought from Colorado, and proved entirely satisfactory and permanent.

It requires about 300 tons (272 m. t.) of sand to form a 26-in. (0.66 m.) bottom for a hearth 19 × 100 ft. (5.8 × 30.5 m.).

The management of a reverberatory furnace in normal action may now be considered, and the Washoe practice may still serve as a type for description.

Assuming the hearth to be filled with its ordinary contents, it will contain some eight inches (0.203 m.) of matte at its deepest point, shoaling gradually as we recede from the tap-hole. On top of this matte floats the layer of liquid slag which varies considerably in depth with the skimming-periods, as well as with the distance between surface of hearth and skim-plate.

The earlier type of these long furnaces was provided with charging-hoppers extending over nearly the entire length of the hearth; but experience soon indicated that better results could be obtained by limiting the charging of the fresh ore to a comparatively small and intensely hot area adjoining the fire-bridge, and extending only about 20 ft. (6.1 m.) therefrom. The hot, roasted ore drops from two hoppers, each provided with two discharge-spouts corresponding to circular openings in the arch. It weighs 15 tons (13.6 m. t.), of which two-thirds is discharged from the hopper nearest the bridge.[2]

[1] "Experiments in Reverberatory Practice, Cananea," read before the Institute of Mining and Metallurgy, Nov. 18, 1909.

[2] This practice corresponds entirely with the laws mentioned on page 313, by which the rate of heat-absorption increases enormously with the difference in temperature between the gases which are doing the heating and the ore which is being heated. The rapidity in this case is almost incredible to the

A fresh charge is dropped at intervals of about one and a quarter hours, the condition of the interior being usually determined by looking through the mica-covered peep-holes of the side-doors. Only occasionally is a door lifted for a moment, and the condition of the hearth explored more thoroughly by means of a rabble. The whole operation of fusion and removal of products in these long furnaces is so totally different from all of our older practice, that a moment's consideration of what is happening inside the hearth will be enlightening.

Before the fresh charge is dropped, the previous contents of the circumscribed smelting-area have flattened out, softened, and begun to merge — to a considerable extent — with the proper liquid contents of the hearth. They are not yet completely liquefied and, if sampled, would show too high values in metal; but they have pretty much completed their chemical reactions, and they are in a condition of mobility. The fresh charge dropping upon this nearly liquefied surface must, of course, displace it to a considerable extent. The bridge and the side-walls bar its movement in three directions, so that it is forced along — floating always upon the liquid material below — toward the distant front end of the furnace. Thus it is gradually displaced and pushed along for some 70 or 80 feet until it reaches the flue-end of the hearth. Although the temperature here is several hundred degrees lower than it was at the fire-bridge, it is still amply high to maintain the slag in its thoroughly molten condition, and this great distance over which it has been slowly impelled gives it every opportunity to satisfy all of its chemical affinities and become thoroughly homogeneous and suitable, as well as to rid itself of matte-prills with a thoroughness that is unattainable in any other manner, and which reduces its copper, gold, and silver values to a point considerably below what one would look for with a matte containing 41 per cent copper, and a slag averaging 43 per cent ferrous oxide.

Fifteen tons of ore dropped every 75 minutes adds rapidly to the contents of the furnace, the production of matte alone amounting to some 60 tons (54.4 m. t.) per 24 hours, for each

practical smelter who is accustomed only to the lower temperatures of former practice. The manner in which 15 tons of fresh ore-charge softens and disappears resembles closely the behavior of a saucer of ice cream in a warm room.

furnace. This is tapped when the converters require it, one or two ladles (8 to 16 tons) being withdrawn at each tapping. The two tap-holes, in opposite side-walls, are near the bridge, and their level is retained by means of a tapping-plate inserted in the side of the furnace. This consists of an uncooled slab of converter copper 24 in. square, and 2½ in. thick (0.61 and 0.063 m.), which is pierced by the tap-hole. It lasts about six months, is easily replaced, and is practically costless, as the copper of which it is formed is regained in the matte.

The converter ladle runs on trucks on a sunken track along the fire-box ends of the reverberatory furnaces. A cast-iron, unlined launder leads, below the floor-level, from tap-hole to ladle, and discharges 7 or 8 tons of matte into the ladle in about eight minutes.

The opening of the tap-hole is effected with comfort and celerity in the following manner, which is, in the main, about the most suitable way for managing the tap-hole of any furnace under ordinary conditions. The orifice is closed in the usual manner by inserting a fire-clay plug on the end of a dolly, and tapping it firmly into place. Then a sharpened iron rod is immediately driven through the soft fire-clay until the lack of resistance shows that its point has reached the liquid matte of the hearth. In time, as the point becomes dissolved, the rod is driven in further that it may always keep in touch with liquid matte. If at any time it becomes frozen and refuses to advance freely, it is knocked out and the hole chiseled free with hand-bars, replaced by an air-drill if the work becomes laborious. As soon as the air-drill reaches the interior, it is withdrawn, and the pointed iron rod takes its place as before. The exercise of reasonable care in the management of this little operation obviates the annoying tap-hole difficulties which were formerly so common in both reverberatory and blast-furnace smelting.

The removal of slag occurs when the body of melted material in the furnace has risen to as high a level as experience has shown to be suited to the most rapid smelting. With uniform conditions, this stage will be reached at approximately regular intervals of time, and at the Washoe works this occurs six times during the 24 hours, or at intervals of about four hours.

In order to facilitate the removal of the slag, the level of the skim-plate is established at a lower horizon than the desired

surface when the furnace is filled. This result is accomplished by building upon the skim-plate a rude dam of sand, or other inexpensive material (at Anaconda, crushed limestone is used), which holds back the slag until the desired level is attained. The removal of the dam gives free egress to the slag, which runs off in a powerful stream, and without aid from the furnace-man, until toward the close of the operation. The skimming of the furnace is not begun, for obvious reasons, until some considerable time after a fresh charge of ore has been dropped, and consumes about 15 minutes. It causes no interruption in the smelting process excepting such as results from the very moderate diminution of temperature caused by the disturbance of the draft by the cold air which rushes through the skim-door into the flue.

The broad, deep stream of slag flows first through a narrow cast-iron settling-trough for the collection of possible matte-globules, and is then shattered into granules by a powerful stream of water which carries it through iron-lined launders to the distant dump. For this purpose, a slightly-flattened eight-inch pipe (0.203 m.) brings the turbid water of the concentrating mill under a considerable head. Care must be taken that the slag does not flow faster than the water can granulate it, or serious explosions may result. The escape of any considerable quantity would also be dangerous. The flow of slag may be stopped by barring the skim-door with a wide rabble, and banking sand against the latter.

The claying or fettling of the furnace, in the former sense of the term, is an operation which is now almost obsolete. Instead of waiting for the slag to undercut the perpendicular side-walls and bridge, and then to fill this cut at frequent intervals with balls of clay and quartz, it is now customary to merge into the hearth the lower edge of the lining-walls, and cover the whole with a gently sloping mass of sand or silicious ore which has, as opposed to the sea of matte and slag, the character of a shelving beach rather than that of a vertical retaining-wall.

When, in addition to this, the composition of the slag is kept uniform, and comparatively neutral as regards its affinity for silica, its tendency to destroy the edge of hearth and walls disappears to such an extent that the side-walls often go for months without repairs, while even the more exposed bridge seldom requires claying and banking oftener than once in four weeks.

The time for this latter operation is determined largely by the condition of the massive iron plates which form the air channel through the fire-bridge, and which are often cooled by the admission of compressed air. When these plates begin to show a red heat, it is an indication that the protecting layer within the furnace is becoming too thin, and that it is time to renew it.

In preparation for this, the hearth must be emptied of its contents so far as possible, and in order to obtain a product free from slag, the latter is altogether removed from the furnace in the following manner. Matte is allowed to accumulate until it reaches the level of the skim-plate, amounting to 250 tons (227 m. t.) or more. The small amount of floating slag is then skimmed clean, and the matte is tapped off as rapidly as it can be used, the converter operations being modified to suit these unusual conditions. After several hours, the hearth is drained dry; the three working-doors nearest the bridge on either side are opened, and the united labor of a large force of men is employed in throwing about 20 tons (18 m. t.) of the crushed Dillon quartz against the bridge and side-walls.[1]

The flue needs occasional slight repairs, the walls and arch at the hot end of the furnace demand partial reconstruction about every eight months, and once in two years the entire roof and side-walls are rebuilt.

As it requires three or four days for a furnace to regain its normal capacity after even minor repairs, it is evident that the importance of reducing their frequency and extent to the very lowest limit can scarcely be overestimated. It is just in this matter that modern practice excels.

At the Cananea plant, Mexico, Gmahling and Shelby have introduced a method of protecting the side-walls from corrosion which has been found advantageous at that smelter.[2]

The accompanying description refers to Fig. 71.

[1] In the *Engineering and Mining Journal* of May 21, 1910, Redick R. Moore describes a method of introducing sand into the furnace which was found rapid and satisfactory. It consisted in employing a four-inch (0.101 m.) pipe with a T on one end. One branch of the T connected with a hopper containing the sand, while the other accommodated a steam-jet. This device throws quartz-sand 25 to 30 ft. as fast as a man can shovel it into the hopper.

[2] "Experiments in Reverberatory Practice, Cananea," by L. D. Ricketts, Inst. of Mining and Metallurgy, Nov. 18, 1909.

The side-walls of the furnace are built with a slight batter, and 5 × 5-in. (0.127 m.) apertures are left in the arch on either side, close to the walls and 18 in. (0.457 m.) between centers. The fettling material consists of fine, wetted silicious ore which is contained in hoppers suspended from a rail running above each row of roof holes. Every day a suitable quantity of the material is dropped through these holes to such extent as appears necessary. It trickles down along the side-walls and builds up in a sloping bank against the slag-line, where it is consolidated with iron tampers. Excepting during the actual charging of the fettling material, the roof-holes are closed with fire-brick and sealed with fine ore.

This method of mending and preserving the walls has been found economical and efficient at Cananea, as it obviates the emptying of the hearth for this purpose, and works up daily 10 to 15 tons of refractory ores more cheaply than can be done in any other manner.

The same process tried experimentally at Garfield was not found advantageous, owing to the formation of extensive silicious "floaters," which caused delay and the use of extra limestone to dissolve them.

It seems to me that the success of the plan may depend considerably upon the care that is taken to consolidate the loose, silicious fettling material so firmly that it stays in place as an integral part of the lining, and is removed only by the actual fluxing action of the slag. If the character of the fettling ore, and of the labor employed in consolidating it, are of such a nature as to accomplish the desired result, the slag will not dissolve more than it can digest.

As a high temperature is the soul of the smelting process, it may easily be inferred that no feature of the entire operation will better repay care, skill, and minute attention to details than the management of the grate.

The first point to consider is the quality of the coal at one's disposition — leaving wood, oil, and other fuels for later discussion.

While chemical analysis is a useful aid, the physical qualities of coal play so important a rôle in its behavior that nothing but actual experiment forms a safe guide as to its practical value.

When it is feasible, this experiment should take the form of a week's trial in some reverberatory furnace already in operation under skilled management, and should be accompanied with the regular test for calorific value, intensity, length of flame, behavior of ash and clinkers upon the grate, heating value of the escaping

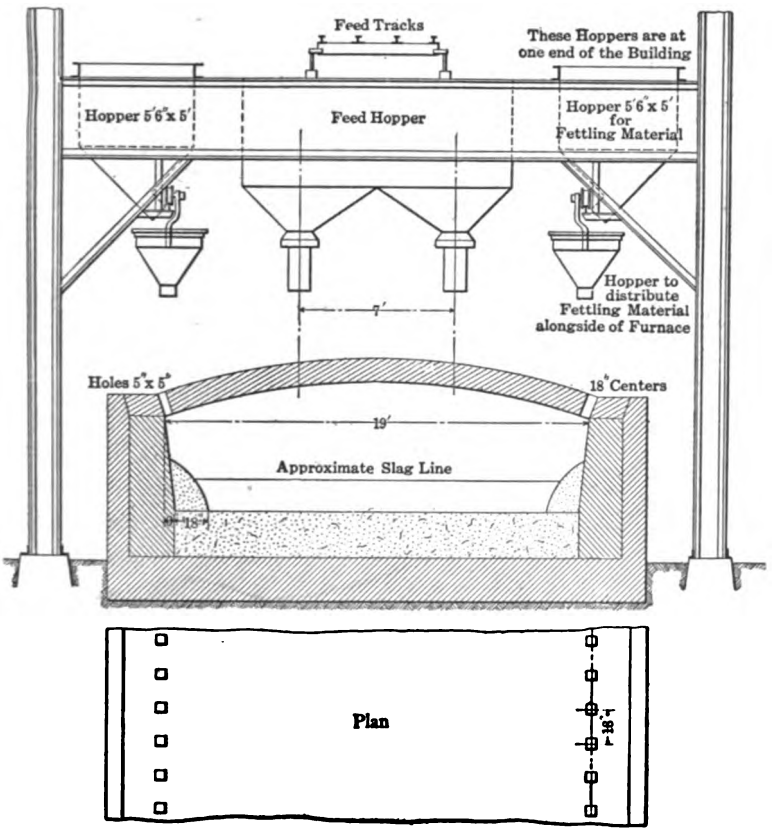


FIG. 71. — Cananea reverberatory. Method of continuous fettling

gases, etc. It is only in this manner that one can determine in advance the many practical questions pertaining to the shape, dimensions, and general construction of the new furnace.

The possible length of the hearth is an especially uncertain factor unless determined by some such experimental work, as of two coals of practically identical chemical composition and appearance and exposed to the same conditions of combustion,

the one may yield a flame which will keep a hundred-foot hearth hot to the very skim-door, while the second cannot maintain a smelting temperature for more than two-thirds of that distance. Other things being equal, the longer the hearth the more complete and economical will be the smelting operation up to a limit of 100, or perhaps 110 feet (30.5 to 33.5 m.). Beyond this point — even if ample heat were still available — it becomes doubtful if advantage enough is gained to compensate for the increased size of buildings and general costs of construction.

In conducting such a test, ample time must be allowed for the furnace-men to become educated to the management of the new fuel. A coal which makes but a poor showing to-day may be doing excellent work to-morrow, and it is a waste of energy to undertake elaborate calorific and pyrometric tests until the men have learned how to use the new fuel to the best advantage, and to maintain a uniform temperature of the maximum grade which the coal is capable of producing.

If the coal will not heat satisfactorily or produce a sufficiently long flame with natural draft, it may become advisable to close the fire-box tightly, install a fan, and try the effect of forced draft. The copper smelter avoids this complication if possible, as it not only causes delay and inconvenience in clearing the grate, but also induces such an intense local heat that the fuel-ash is likely to melt into massive clinkers of tough, aluminous slag which block the grate, disorganize the fire, and are most difficult to break up and remove.

The analyses and descriptions of a few coals which are, or have been, in regular use at these long reverberatories will serve as illustrations.

The Washoe smelter is fortunate in having a coal peculiarly suited to its needs. It is a semi-bituminous coal mined at Diamondsville, Montana, and has the following average composition:

water	6.9	per cent
volatile matter	41.4	" "
fixed carbon	41.0	" "
ash	10.7	" "
	100.0	" "

When dry, it has a thermal capacity of 12,135 B.t.u. per pound. The waste gases from this coal escaping at the end of

the 115-ft. hearth have a temperature of 1200 deg. C., which falls to 950 deg. C. by the time it reaches the boilers, and develop continuously 573.4 H.P. per furnace.[1]

Extensive tests have been made at this same plant on a cheaper coal from Bear Creek, Montana, which has the following composition:

water	13.0	per cent
volatile matter	38.0	" "
fixed carbon	38.0	" "
ash	11.0	" "
	100.0	" "

When dry, this coal has a value of 11,470 B.t.u. per pound. Its waste gases, under conditions identical with those in the preceding illustration, developed 580 H.P. per furnace.

The behavior of these two coals during combustion varied much more than their analyses would appear to warrant. The Bear Creek coal, with natural draft, was unable to maintain the desired heat at the flue-end of the long hearth. With underwind this evil was remedied, though not without decided increase in fuel-consumption.

At Cananea, Mexico — before the introduction of oil as a fuel — two varieties of coal from New Mexico were used, neither of which could satisfy the exacting demands of the 100-ft. (30.5 m.) reverberatory hearths.[2]

One of these coals was a high-ash, strongly coking coal from the Dawson fields. With the powerful draft inseparable from the production of so long a flame, the volatile matter burned off rapidly, leaving upon the grate a heavy bed of slow-burning coke. Its composition was

water	1.7	per cent
volatile matter	32.6	" "
fixed carbon	52.4	" "
ash	13.3	" "
	100.0	" "

The dry coal corresponded to 12,135 B.t.u. per pound.

[1] These results of exhaustive tests were kindly furnished me by the Research Department of the smelter.

[2] See paper by Ricketts, already cited.

The second coal was a good lignite from the Gallup basin, with the following analysis:

water	10.7	per cent
volatile matter	34.6	" "
fixed carbon	48.4	" "
ash	6.3	" "
	100.0	" "

B.t.u. per pound dry coal, 10,247.

This coal was too light; it burned too much like wood, and did not produce the required intensity.

On the whole, a half and half mixture of the two coals yielded the most satisfactory results, although they were but poor at best.

An example may be added of a coal high in volatile matter, and the experience at Cerro de Pasco, Peru, at an elevation of 14,000 ft. (4267 m.), will be interesting.[1]

These furnaces have a hearth of the usual width, but a length of only 60 ft. (18.3 m.), and a grate 7 × 19 ft. (2.1 × 5.8 m.).

The analysis of the coal shows

water	6.2	per cent
volatile matter	39.0	" "
fixed carbon	33.7	" "
ash	21.1	" "
	100.0	" "

The high proportion of volatile matter, together with the great elevation of the plant above sea level, demands unusual space for combustion of the gases; consequently, a clearance of 41 in. (1.041 m.) was left above the bridge, and a flue-opening of 25 × 84 in. (0.635 × 2.13 m.) was constructed. The smallest section of the flue proper was 50 × 54 in. (1.27 × 1.37 m.).

Forced draft is used, each of the five furnaces being equipped with an electrically-driven No. 9 Sturtevant blower making 1000 revolutions per minute. As is so frequently the case, the forced draft melted the ash of the coke to the sides of the fire-box so extensively that the labor of grating became insupportable. This difficulty was overcome by protecting the bridge-wall of the fire-box with a water-jacket, and extending the latter across the

[1] See paper in Bulletin 67 of the Institute of Mining and Metallurgy, by S. T. Gormly, superintendent of the Cerro de Pasco smelter.

entire width of the furnace, so that the feed-water and overflow connections were outside. The addition of a tilting grate, by which one-half of the entire contents of the fire-box could be dropped at once, completed the installation, and reduced the grating period to 15 or 20 minutes.

These furnaces now average about 125 tons (113.4 m.t.) of ore daily, on a fuel ratio of 1: 2.75, producing a 35 per cent matte, and a slag with:

SiO_2	36.0	per	cent
FeO	41.8	"	"
CaO	5.3	"	"
Al_2O_3	3.1	"	"
MgO	0.4	"	"
PbO	2.1	"	"
ZnO	3.4	"	"
S	0.3	"	"
Cu	0.42	"	"
	92.82	"	"

The management of the temperature in a reverberatory smelting furnace is regulated by the ordinary laws of combustion where a high temperature is desired.

The main duty of the fireman is to burn coal as rapidly as possible, and the details by which this purpose may be accomplished are not peculiar to copper metallurgy, and vary materially with the character of the coal.

In a general way it may be said that lump coal can be burned in a deep layer and fed at long intervals, while fine coal requires a shallow bed and frequent firing. A coking and baking coal demands frequent breaking and stirring, while a dry, free-burning fuel may be left undisturbed, as its air-passages do not tend to choke.

The clearing of the grate from ash and clinkers is a matter demanding especial care and judgment, and nothing but local experience can determine whether it may best be conducted as a more or less continuous process — by which the clinkers are broken up and hooked through the grate-bars at frequent intervals; whether it is best to allow the mass to accumulate, and practise a thorough grating every few hours; or whether, as at Cerro de Pasco, it will be necessary to dump completely one-half of the fire at certain regular intervals.

Complete combustion is not desired in the fuel-chamber. Our object is to melt the *ore*, and not the brick walls of the fire-box; and the ore is situated in the hearth compartment. The fire-box is, in great measure, a gas-producer in which the volatile portion of the coal is distilled and the carbon is burned mainly to CO. These intensely hot gases, hungry for oxygen, mix (imperfectly) with the air rushing in through the roof-holes above the fire-bridge, and burn with a very high temperature. But combustion is yet far from complete; if it were, there would be but a poor supply of heat for the remainder of the long hearth. Gases, under the conditions here existing, do not mix with anything like the thoroughness and rapidity that one would expect, and analyses show clearly that three distinct sets of gaseous substances exist at the bridge-end of the hearth. It may be fanciful to attempt to divide them into horizontal layers; but, for illustrative purposes, I shall venture to describe them as existing in far more separate and distinct strata than is actually the case.

We may, then, imagine the heated gases at the bridge-end of the reverberatory hearth as consisting of (*a*) a layer of unconsumed air hugging the under surface of the arch; (*b*) a deeper layer of the products of combustion, which are inert and passive, and diluting; (*c*) a bottom layer of active gases, anxious for oxygen, and comprised mainly of CO and hydrocarbons. These layers are, of course, most imperfectly defined, and are constantly mixing as they roll along in waves through the hearth-space; (*a*) and (*c*) constantly diminishing in quantity (and constantly generating heat as they diminish), and (*b*) as steadily increasing in volume. The layer of active gases (*c*) cannot develop its power until supplied with oxygen (*a*), and this supply must be furnished in suitable quantity and at the proper moment. When everything is in normal train, the gradual mixing of (*a*) and (*c*) will continue along the entire extent of the hearth, and will be completed when the gases reach the skim-door and turn upward into the flue. If (*a*) is completely exhausted before that moment, unburned (*c*) will pass into the flue, causing excessive heat when it burns in that region, and a corresponding loss of fuel.

This is well exemplified in practice when the smelter finds the far end of his hearth becoming cold. The remedy is to supply the cold end with more of the unburned gases (*c*); and, to accom-

plish this, he shuts off a portion of the air from the grate-end of the furnace, and thus throws a mass of unburned gases toward the distant portion of the hearth; indeed, the flue could easily be melted down by carelessness in this respect. In cases where it is impossible to force enough unburned gases toward the far end of the furnace to maintain the necessary temperature, it is evident that the hearth is too long for the quality of the coal.[1]

At the Washoe smelter, coal is charged from a row of four hoppers delivering through holes in the roof of the fire-box which are closed with sliding, rectangular covers. Three thousand pounds (1361 kg.) of coal is charged every 40 minutes, or a smaller quantity oftener, if the coal is too fine.

The condition of the flame is the main criterion of the combustion, and is observed through a mica-covered peep-hole situated in the wall of the flue at the far end of the hearth. The checker-holes in the arch are opened or closed in such a manner as to preserve a certain amount of flame at the flue-end of the hearth.

Grating, at the above smelter, is necessary at intervals of about three hours, the clinkers being broken up with heavy bars, and a certain proportion of them hooked down through the grate spaces, while the rest is manipulated so as to preserve a rude bed for the fire. This work is hot and fatiguing and takes about 40 minutes to complete. About seven per cent of the total coal charged is recovered from the ashes, as already described, and is used in the blast-furnaces.

Careful determinations at Anaconda, representing normal conditions, indicate that the temperature above the fire-bridge is about 1550 deg. C., while that of the escaping gases at the flue-end of the hearth averages about 1200 deg. The temperature of the slag as it leaves the furnace is 1120 deg. C., which drops to 1060 deg. C. at the overflow spout of the settler.

The temperature of the gases where they reach the boiler is 950 deg. C., and about 330 deg. C. on leaving them.

An interesting test to determine the final destination of the heat produced in these furnaces gave the following result, showing that it was possible to account for 78 per cent of the heat value of the coal burned.

[1] I am, of course, assuming that the furnace is properly proportioned and has sufficient draft. A badly designed reverberatory may fail to heat even with the best of fuel.

HEAT BALANCE

heat in slag	16.34	per	cent
heat in matte	3.18	"	"
heat loss by radiation	11.55	"	"
heat loss in cooling bridge-plate	0.16	"	"
sensible heat in grate-droppings	0.78	"	"
heat in steam generated in boilers	32.81	"	"
heat in gases after passing boilers	13.18	"	"
	78.00	"	"

At Cananea 34.8 per cent of the heat generated was recovered in the boilers, 36.5 per cent being utilized in the actual smelting operation.

The question of draft stands in intimate relation to that of combustion. This is one of the vital points connected with the operation of the reverberatory furnace, and should have the most careful consideration of the designer.

I have never found that theoretical calculations as to the size of the flue and chimney required for a given situation were at all satisfactory as a basis for actual construction. There are too many disturbing factors, any one of which may neutralize all the refinements of the most elaborate calculation. If we had nothing more tangible than these to guide us, we should of course have to use them and make the best of the situation; but with the mass of practical illustrations spread before us, we may find examples to suit almost any reasonable conditions, and I advise strongly that all theoretical calculations be examined and corrected with the aid of a thorough study of successful examples.

The demand of the individual furnace is now almost always lost in the mass of air-currents which converge in a central stack, and the engineer who desires to construct a furnace with individual stack will find it difficult to determine what minimum size and height will suffice. The matter is still further obscured by the countless variations of the surrounding conditions, such as height above sea level, the influence of adjacent hills or valleys, the quality of the fuel, and other less striking influences, which combine to render futile any attempt at close calculations. The safest and most reasonable plan is to study approximately similar situations, and then to allow a liberal margin for safety. In the United States and Mexico, most of the great reverberatory smelting plants are situated in mountain or plateau regions at eleva-

tions above sea level varying from 4000 to 8000 ft. (1219 to 2438 m.). The rarified air demands more space, and all furnace passages must be from 10 to 40 per cent larger than at sea level.

To speak first of moderate-sized furnaces with individual stacks, I have selected about 30 of such furnaces which are, or have been, doing satisfactory work on a fair grade of semi-bituminous coal, and find that, after throwing out a few excessive variations, the ratio between areas of stack and grate averages almost exactly 1:1.9; or, in other words, that each square foot of grate is provided with 0.526 sq. ft. of chimney area. These furnaces will average about 800 sq. ft. of effective hearth area, and will smelt per 24 hours about 250 lb. of hot, fusible, roasted ore per sq. ft. of hearth area (1220 kg. per sq. m.), with a ratio between ore and fuel of 3.2 : 0. The average height of the separate stack belonging to these furnaces was 72 ft. (22 m.), there being little variation in this dimension. Increasing the height of the chimney beyond this point was not followed by increase in capacity (3 cases). The weight of coal burned per hour for each sq. ft. of stack area ranges from 70 to 100 lb. All of the statements in this paragraph refer to furnaces situated at 5000 ft., and more, above sea level.

At the present time we pay much more attention than formerly to determining the exact velocity of the gases through the hearth, and this is particularly important with our large reverberatories which are connected with one great central stack.

The measurement of the draft-pressure within the furnace or flues may, however, be quite misleading unless it covers considerable periods of time, or is repeated frequently. Large numbers of such determinations are accessible, covering many of the great and medium-sized reverberatories of this country; but I believe it will be safer to confine our examination to a lesser number of more continuous and repeated results.

As usual, the Washoe smelter, with its eight large reverberatories, and its organized department of research, furnishes the most continuous and extensive records. At one of its 115-ft. (35 m.) furnaces, the average results for 30 days were: [1]

	neg. pressure
draft at bridge-wall	0.88 inches
draft in flue after passing boilers	1.89 "

[1] The negative pressure is expressed here in inches of water. (One inch = .0254 m.)

At the Steptoe valley reverberatories, as well as at those of the Garfield and Cananea plants, a similar negative pressure is maintained. Any considerable decrease in the same at once reduces capacity.

The accompanying autographic record of the draft-pressure at one of the Anaconda furnaces for one shift shows how such measurements may be made with little manual labor and, taken in connection with the table entitled: "Memorandum of furnace operations during one shift," indicates the effect which these operations produce upon the draft, and constitutes also an excellent check upon the furnace-men.[1]

The almost complete momentary checking of the draft from the dropping of the ore-charge will be noticed, as well as the lesser disturbance due to the dropping of the coal.

TABLE 2. — MEMORANDUM OF OPERATIONS DURING ONE SHIFT

Skimming	7.00–7.14
(Coal charged during skimming.)	
Side door opened	7.16–7.16½
Coal charged	7.28–7.31
Charged	7.52–7.57
Tapped	8.05–8.15
Coal charged	8.15
Coal charged	8.40
Grating	8.43–9.17
Side door opened and charged	8.54–8.59
Coal charged	9.05
Coal charged	9.27
Coal charged	9.49
Charged	10.07
Coal charged	10.25
Coal charged	10.41
Skimming	10.45–10.58
Coal charged	11.04
Charged	11.16
Some grating	11.16–11.35
Coal charged	11.35
Coal charged	12.03
Charged	12.04
Tapped (1½ ladle)	12.37–12.48½
Coal charged	12.45

[1] For this record and table, as well as for several other details, I am indebted to an excellent and reliable article by C. Offerhaus, published in the *Engineering and Mining Journal* of June 13 and 20, 1908.

Charged	1.00
Grating	1.11–1.45
Coal charged	1.26
Charged	1.44
Coal charged	1.51
Coal charged	2.18

Total charges made:
16 coal charges.
7 calcine charges.

In modern plants the arrangements for delivering coal and ore into the furnace, and for disposing of the slag, matte, and

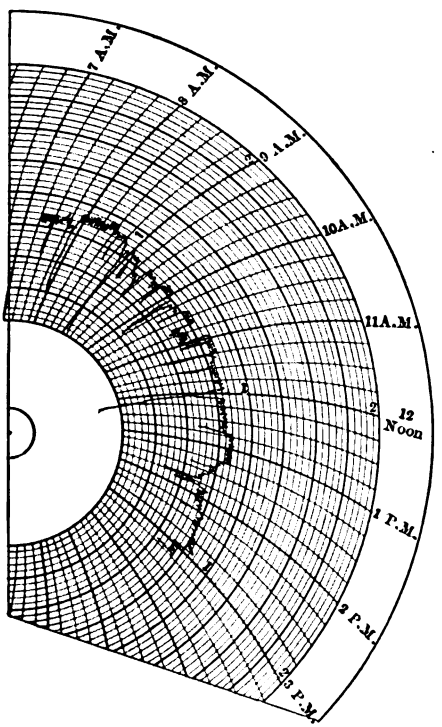


FIG. 72. — Washoe reverberatory draught-pressure

flue-dust, are so carefully planned that the amount of labor required is surprisingly small; and much of this labor is really supervision and regulation rather than actual toil.

The large Anaconda reverberatories require one skimmer and

two helpers on each eight-hour shift — this being nine men per 24 hours, or 72 men per 24 hours for the eight furnaces. In addition there is one man on each shift to watch the feed-water for the boilers; two men on each shift to dump the cars of hot roasted ore; and one man on each shift to dump the cars of coal; also two men on day shift only to repair the various launders. Omitting foremen, this makes a total of 86 men for the 24 hours, or about 23.25 tons (21.08 m. t.) ore smelted per man-shift, which is unusually favorable.

While the details of management differ somewhat at the various reverberatory plants with their differing ores and fuels, there are certain general principles that are common to all, because they are based upon universal laws of physics, chemistry, and economics.

Bearing always in mind that the reverberatory is a costly furnace, designed for producing certain peculiar results which cannot possibly be attained in any ordinary apparatus, it follows that it should be dedicated exclusively to these dignified purposes, and should not have its time and energy frittered away on any class of work that can be accomplished equally well by a less highly organized agent.

We have demonstrated already that the primary duty of the reverberatory smelting furnace is to produce a high temperature, and to provide a secure bed upon which the liquid products resulting from the proper utilization of this unusual heat may collect and separate.

We have further examined the means at our disposal for maintaining and conserving this rare and costly temperature, as well as for making the repairs necessary to the existence of the apparatus in which it is evolved.

We have yet to consider certain general points which experience has taught are necessary for the most advantageous application of this definite number of heat units which we have at our disposition.

For reasons specified in the introductory pages of this chapter, the reverberatory furnace is essentially an apparatus for the smelting of finely-divided material — often termed fines — and any considerable admixture of lump ore delays its normal operation. It is evident that we must not call upon a costly furnace to sur-

mount a difficulty which can be obviated by employing a cheap crusher; so one of the first questions that would occur in this connection would be — what degree of fineness is essential to the most rapid and economical fusion?

The reply to this question depends mainly upon the character of the ore. If silicious, it will be hard to melt and will require fine crushing, and this condition will be accentuated if it contains its silica in patches of any considerable size. Contact is the essential feature for rapid reverberatory smelting of refractory substances, and this condition can be most easily fulfilled by fine crushing and thorough mixing, so that each grain of quartz may — so far as possible — lie in contact with a fragment of basic material, so that it may begin to soften into slag at the earliest possible moment. If distinctly basic — and, especially, if the bases are the easily fusible iron or manganese oxides — lumps of roasted ore as coarse as two in. (0.051 m.) in diameter may be used without materially delaying the process.[1]

As vague generalities, however true, are of little use to the practitioner, I will attempt to lay down certain rules for the benefit of those who have not personal experience to guide them. I believe that a highly silicious ore, or an ore containing much massive zinc blende (such as certain silver-bearing ores sometimes added to copper furnaces), should be crushed as fine as one-fourth inch (0.006 m.) in order to ensure rapid fusion. Roasted lump ores (which, naturally, contain a good deal of iron, and which tend to swell and crack during the roasting) should, if silicious, pass a one-half inch (0.012 m.) screen; while basic, roasted lump ores may, as a rule, be used in fragments as large

[1] In smelting stall-roasted, first-class, pyritic lump copper ore from the Parrot mine, at Butte, in 1883, I attempted to shorten the (wood-burning) reverberatory work by crushing the roasted ore through a three-quarter inch (0.019 m.) screen, but found no perceptible gain in time. The quartz was well disseminated.

In 1870 Hermann Beeger, metallurgist of the Boston and Colorado smelter at Black Hawk, told me that, at one time, when their crushing-plant was closed for repairs, the omission of the crushing of their heap-roasted, lump ore (Gilpin county copper-gold pyrite, of a silicious nature) curtailed the capacity of their wood-burning reverberatories by nearly one-third. This was due to the fact that the lump ore carried its silica in segregated form, and that these lumps formed a silicious mass upon the hearth that required persistent firing and rabbling to remove.

as 1½ inches (0.038 m.) in diameter, or even larger, provided their silica does not occur in too segregated a form. Of course the reverberatory, as a rule, is used only for ores which are already in a condition of considerable fineness.

In close relation to the question of size of particles stands that of uniformity of mixture. It is impracticable to bring about the ideal condition, in which each particle of silica should lie next to a suitable particle of base; but the nearer this situation can be approximated, without entailing too much expense, the more satisfactory will be the smelting operation. With the lower temperatures of the older furnaces, any carelessness in the matter of mixing the charge became evident in a most emphatic manner, especially in custom smelting, where various small lots of differing, and often highly silicious, ores go to make up each charge. The head furnace-man usually took good care that there should be no shirking in the mixing department, knowing else that it would cost him an hour or two of extra firing and rabbling to get a clean bottom.

With the higher temperatures and the matte-protected bottom of the present day, any shortcoming in the mixing may escape detection unless it is made the subject of especial investigation; yet the injury is as real as in the older practice. It limits the capacity of the furnace by making it more tedious and difficult than necessary for the various constituents of the charge to combine to form the slag which is the eventual object of the operation.

At works which purchase outside ores, this matter of mixing becomes so important that especial measures must be taken to secure reasonable uniformity of the charge. An obvious measure is to pass the various lots of ore together through the roasting furnace, by which a thorough mixing is effected, provided reasonable arrangements are established for feeding the different ores into the roasters, so that they may not be too much isolated.

At the Garfield smelter, where the Utah Company's concentrates (which carry 20 per cent of copper, so that they really require no roasting) are added on the fifth (next to lowest) floor of the McDougall roasters which are already engaged in roasting ordinary concentrates, an addition of 15 tons (13.6 m. t.) daily of the rich Utah concentrates is possible, while still roasting 40 tons (36.3 m. t.) of the lower-grade sulphides. By this plan,

the rich concentrates are dried, preheated, and uniformly mixed, while the flue-dust is diminished.

At Anaconda, and elsewhere, the crushed limestone needed for flux is fed with the sulphides into the McDougall roasters. This preheating and uniform mixing aids the smelting process materially without diminishing the capacity of the roasters, as the refractory limestone permits a higher heat during the earlier stages of roasting.

Actual bedding and reclaiming of the ores before entering the roasters may be advantageous in a custom-smelter.

Apart from the inconvenience and serious danger of dropping moist ores onto the hearth of a smelting furnace, the mere evaporation of the moisture abstracts a great quantity of heat from the very place where it can least be spared.

As the material smelted in most reverberatories consists mainly of roasted sulphides, or of flue-dust, the metallurgist is seldom troubled with too moist a charge, the greater portion of it, indeed, entering the smelting-hearth at a temperature of several hundred degrees. In cases, however, where fine, wet, unroasted material has to be added, some provision for drying it must be made. Moore points out [1] that the McDougall roaster forms an economical and efficient drier, and that it is used for this purpose at the Garfield smelter, where it dries 60 tons (54.4 m.t.) per day of silicious ore to serve as flux in the basic-lined converter, with the consumption of only 300 lb. (136 kg.) of Utah coal. This result was accomplished by changing one of the doors of the bottom floor into a small fire-box, and causing all the air that entered the furnace to pass through this door.

The removal of the slag and matte has been already discussed; one further point may be noted regarding the latter substance.

The so-called "boiling" of the matte is now a rare phenomenon in reverberatory smelting for the simple reason that, in our present practice, this substance is kept covered almost continuously by a tolerably thick layer of slag, while the slag is kept from excessive overheating by the frequent dropping upon its surface of fresh portions of comparatively cold ore.

The boiling shows itself when there has been persistent firing upon a naked bath of matte and, as indicated by its name, con-

[1] *Opus cit.*

sists in a strong ebullition, accompanied by the escape of countless bubbles of gas. This gas appears to consist entirely of SO_2 and SO_3, the latter being present in far greater proportion than we are accustomed to expect from the oxidation of sulphides.

Owing to the fact that several independent observations have shown that continued boiling lessens the ratio of sulphur to iron in the matte, and yields matte which contains more iron than can be accounted for according to our customary theories (and which is not present as Fe_3O_4), I think it is reasonable to assume — in default of any better-proven explanation — that when the persistent superheating has brought the matte to a sufficiently elevated temperature, the FeS begins to become dissociated, Fe remaining dissolved in the matte while S is distilled in its elemental form, oxidizing to SO_2 and SO_3 as soon as it reaches the surface.[1]

Whatever may be the cause of matte-boiling, its effect is dangerous, as it tends to tear up any weak portions of the hearth, as well as to disorganize the fettling and work its way down, or through, the side-walls and bridge.

The employment of pulverized coal as a fuel for copper furnaces has attracted considerable attention of late years.

The advantages of this system were too well known and too thoroughly demonstrated (in the cement industry, and elsewhere) to require defense. Dust-firing approximates gas-firing. There is an intimate mixture of the substance to be oxidized with the agent employed for the oxidizing, and thus complete combustion is obtained with a minimum quantity of air and a maximum evolution of heat, while the exact point at which the combustion shall take place is under the control of the operator.

Appreciating fully all of these advantages which would result from the application of dust-firing to his reverberatory furnaces, and which would convert coal fines and sluggish-burning coals into an eligible fuel, — so far as the mere production of heat is concerned — it remained for the copper smelter to determine what disadvantages would become manifest when he subjected this system to every-day conditions at his furnaces.[2]

[1] See "Principles of Copper Smelting," Chapter X, for a detailed study of the composition of matte.

[2] Some of the details given herewith are taken from papers by S. Severin

A brief account of the actual results obtained by experiments conducted on an extended scale at two important plants will present the subject in a form more useful for technical purposes than any general deductions are capable of doing.

The reverberatory furnace of the Highland Boy (Utah) smelter at which these tests were conducted had a tapering hearth 38 ft. long (11.58 m.), with a fire-box 7.5 × 8 ft. (2.29 × 2.44 m.), and carried a bed of coal three feet deep on the grate. The 80-ft. (24.4 m.) stack was insufficient to create the required heat with the coal available for use, so that forced draft and a closed ash-pit were necessary. This was accompanied by the usual annoyances of clinkered ash, choked grate, and laborious chiseling-out of the same at frequent intervals and with the loss of so much time and heat that it became necessary to attempt some improvement.

It was decided to try dust-firing and a centrifugal impact pulverizer was installed, requiring 30 H.P., and making 1400 R.P.M. The coal-dust produced by this machine was exceedingly fine, 90 per cent of it passing through a screen with 150 meshes to the inch (1 in. = 0.0254 m.). By means of a fan, the coal-dust was blown into the furnace a little above, and at the rear of, the bridge, through a single flat, flaring nozzle, about one ton (0.907 m. t.) per hour being used. Two causes of trouble were anticipated: (*a*) possible explosions; this was guarded against by water-jacketing the nozzle, and shutting off the coal-feed before stopping the pulverizer; (*b*) ash from the burned coal-dust forming a protecting blanket over the surface of the charge. This did not occur to any serious extent, and the small quantity that gathered in flue and stack was barred off easily. The flame was amply long, and the heat could be regulated so as to maintain the desired temperature at any portion of the hearth.

A considerable amount of delay and annoyance was experienced which, for a time, threw some discredit upon the experi-

Sörensen, Engineer of the Utah Consolidated Mining Co., *Engineering and Mining Journal* of Feb. 10, 1906; also an article by Charles F. Shelby, Superintendent of the Reduction Division of the Cananea Consolidated Copper Co., *Engineering and Mining Journal* of March 14, 1908. These statements have been checked and supplemented by personal details from other sources to such an extent that I am unable to give credit to every person who has aided me in the matter.

ment in the eyes of the public; but the testimony of Mr. Sörenson, as well as that of several of the workmen who were employed there, seems to indicate that the cause of these different troubles was, in every instance, removable, and that no fundamentally important obstacle was encountered. The main troubles arose from excessive moisture in the coal, and from certain mechanical difficulties which had no direct relation to the process itself. The results of two months running indicated that — after deducting the coal used to drive the pulverizer — the dust-fired furnace showed a saving of from 15 to 20 per cent over the other and larger reverberatories.

The results were regarded as satisfactory to the management, and plans were under consideration for applying the same system to all of the other furnaces; these, however, were never carried out, owing to reasons foreign to this subject.

The second illustration of the results obtained by the use of pulverized coal comes from the Cananea (Mexico) smelter.[1]

The only available coals being unsatisfactory for reverberatory work, the fire-box of the furnace was removed, pulverizers installed, and dust-firing undertaken. The conditions of flame and temperature became at once much more satisfactory, but grave difficulties also began to appear, and demanded much experimenting before any reasonable results could be obtained. The most serious and persistent of these troubles was the steady choking of the flues with the fine particles of ash resulting from the burned coal-dust. These particles could not be treated and removed like ordinary flue-dust, as they were semi-viscous soon after deposition, and hardened quickly into a solid clinker extending fifty feet or more from the throat of the furnace.[2]

Mr. Shelby reports that the mixture of coal-dust and air was blown through the rear wall of the furnace by means of two flattened nozzles, each 2 in. by 50 in. (0.51 × 1.27 m.) with supplementary air-slots. The round nozzles used at the start failed to secure efficient combustion, and caused the deposition of a layer of coke over the charge, which insulated it from the heat.

When the furnace was changed from ordinary firing onto pulverized fuel, its capacity increased rapidly from its previous 100-ton (90.7 m. t.) level to 180 tons, when it began to fall as

[1] See page 341 for analyses of coals, and description of conditions.

[2] *Opus cit.*

rapidly as it had risen. This decrease was due to the deposition of ash in the flues already referred to. Finding that, under the new system, the boilers were yielding no adequate results, a low stack of 24 sq. ft. (2.23 sq. m.) area was built directly at the throat of the furnace. This was intended merely for experimental purposes, and remedied the trouble with the ash in the flues.

The next difficulty requiring attention was the deposition of a refractory, silicious blanket of ash upon the surface of the ore-charge. Each charge consisted of 15 tons (13.6 m. t.) which was dropped upon the 25 ft. (7.62 m.) of hearth adjoining the bridge, and which had ample time — before it became melted — to receive its insulating layer of ash. This trouble was remedied by a system of almost continuous feeding, by which 800 lb. (363 kg.) of charge was dropped at intervals of three to four minutes, thus breaking up and fluxing the layer of ash before it reached an appreciable thickness. The charging-holes, not being constructed for such continuous work, soon became slagged out of shape, and so much time was consumed in repairing them, and in other changes, that it was thought better to stop the furnace and put it into thoroughly good condition, with the addition of such improvements as experience had suggested.

Just at this time the Mexican government modified the duties on fuel-oil in such a manner that there could be no doubt as to the advantages that would result from its employment at Cananea, and the projected equipment for dust-firing was abandoned permanently.

I feel, however, that the dearly-bought experience resulting from this test-run is too valuable to the profession to remain in its present somewhat inaccessible form, and will sum up briefly some of Mr. Shelby's plans for refitting the plant in the light of his recent experience.

He intended to cut off the fire-box of the reverberatory completely, and build a wall up to the roof, just back of the fire-bridge, piercing it with apertures for the feed-nozzles for coal and air. The throat of the furnace was to be enlarged to about double the area of the former Anaconda standard. The small stack was to be used temporarily until the experimental stage had been passed completely. The charge-holes in the arch were to be lined with iron pipes of 2 in. (0.051 m.) inside diameter, and with water-cooled walls three inches (0.076 m.) thick cast around

a spiral-pipe coil. The bottoms of these pipes were to be flush with the under surface of the arch, while the upper ends would project about 12 in. (0.305 m.), and were to make close connection with the feed-hoppers, to avoid the spilling of ore. They were to have sliding doors of fire-clay tile, or water-cooled iron.

It will be noted that in firing copper furnaces with coal-dust little, if any, complaint is made as to the one fundamental purpose for which all this firing is done: namely, the production of heat. So far as I am aware, every serious difficulty that has arisen has come from what might be classed among mechanical details: being, for instance, insufficient power or faulty arrangement of coal-pulverizers; incomplete combustion of the coal-dust, due to imperfect admixture of air; mechanical difficulties due to settling of the ash; unsatisfactory methods of ore-charging, etc.

All of these points have been ameliorated, and can be, in all reasonable probability, completely overcome. Most of them need never have been encountered had the experimenters made use of the knowledge and skill already acquired in the application of this method to other branches of industry.

It is scarcely necessary to state that the employment of pulverized coal to produce temperatures of 1400 deg. C. and upwards has been an established practice for a considerable number of years, and there seems no reason why it cannot be applied successfully to copper smelting.

The whole subject is complicated by having to consider more than the mere smelting of the ore. No matter how advantageous we find it to employ pulverized coal as a fuel, nor how skilfully we may manage the production and application of the heat arising from its combustion, we shall still have left on our hands a large volume of highly heated gases. As soon as the temperature of the gases passing through the furnace becomes reduced to a point anywhere near as low as the temperature of the slag at the skim-door, they have lost their value in the smelting process, and must be removed to make way for fresh accessions from the more heated region. In plainer words, there must be draft enough to keep the hearth-gases moving with such a speed that there shall always be sufficient heat at the skim-door.

This means that we have continuously on our hands a great

volume of gases heated to 1000 deg. C., and upward. In most of the modern reverberatory work it has become the custom to utilize a considerable proportion of the heat in these gases to make steam, and several independent tests lead us to believe that about one-third of the total heat-value of the coal burned in the furnace grate is thus recovered in the shape of power, amounting in a full-sized furnace to 500 or 600 continuous horse-power. This is an exceedingly important item on the credit side of the smelting costs, and might easily turn the scale in selecting the process for a new enterprise.

We find, however, that in dust-firing the flues and boilers become so choked with the resulting ash as to interfere seriously with the legitimate function of the smelting furnace, and it has even been suggested that we throw over the utilization of the waste heat for power, and content ourselves with the advantages resulting from the improved smelting. If it were manifestly impossible to utilize this waste heat while employing dust-firing, the proposition might be a sound one. Instead of the expensive boiler-installation, our pulverizing-equipment would be comparatively trifling, and it would only remain to determine whether the saving effected by the use of pulverized coal would more than compensate for the loss experienced by giving up our power plant.

I do not think, however, that we are likely to be reduced to any such choice as this. Coal-dust fired boilers, with temperatures in fire-box of over 1400 deg. C., are used extensively and without serious difficulty. Puddling-furnaces provided with waste-heat boilers are fired satisfactorily in the same manner, and cement furnaces — showing pyrometric tests of 1500 deg. C. — sometimes employ coal-dust high in ash.

The problem, no doubt, is a difficult one when added to all the other complications of experimental work under adverse conditions; but taken by itself, and with favorable surroundings, there can be no question that it is capable of solution.

The general summing-up of the situation at present seems to be that, while pulverized coal gives excellent results as regards economy, intensity of temperature, length of flame, and general ease of management, it is accompanied by a number of serious difficulties, none of which appear fundamental or insurmountable; and that, in order not to have to commence at the beginning

each time, it is advisable to employ an expert in coal-pulverizing and dust-firing, and allow the copper smelter to devote his undisturbed energies to removing such obstacles as are introduced by his own peculiar conditions. The most pressing application of dust-firing would seem to be in those cases where an inferior coal may, by this modification, be converted into an acceptable fuel for reverberatory smelting.

The employment of fuel-oil for the production of heat has been an established practice for so long a time that its application to copper reverberatory smelting furnaces presents no radically new features. As with coal-dust firing, such obstacles as have been encountered have been due largely to the inexperience of the operators, and to certain mechanical defects which have been overcome without unreasonable difficulty.

The great advantage which oil possesses as a fuel is that it burns completely to gaseous products, and thus leaves no residuum or ash. This property alone relieves the smelting process to an extent that can be appreciated only by the practical smelter and which, if it could be expressed in dollars and cents, would probably reach a larger sum than even he would estimate. The expense of the oil is the one serious drawback to its employment in copper smelting; but where the plant is so fortunate as to be situated where this objection does not exist, I think there can be no question as to giving it the preference over all other fuels excepting natural gas.

As might be inferred, there is little trouble in maintaining a suitable temperature with such an admirable combustible as fuel-oil. The main difficulties which have been encountered relate to the control and distribution of this heat, to the manner in which the oil should be burned to obtain the desired effect with the greatest possible economy, and to the maintenance of the furnace in which this heat is to be produced and utilized.

The most suitable manner in which to study the practical side of the subject will be to examine some of the copper plants which have adopted oil as a fuel, giving particular attention to the obstacles which were encountered, and to the means by which these obstacles were overcome.[1]

[1] I shall make free use of the following articles, all of which I recommend to those interested in the subject. This general acknowledgment may take

The most exhaustive and successful experiments with which I am familiar in the use of oil as a fuel in reverberatory copper smelting furnaces are those lately carried out at the Cananea smelter. It is fortunate for the profession that this work was under the direction of engineers able and willing to communicate their results to the public and that, having attained complete success, they could the more fully describe the most instructive portions of their work: namely, the difficulties encountered and surmounted.

The conditions at Cananea have been described in preceding pages while discussing the subject of firing with pulverized coal. It must suffice here to say that the 19 × 100 ft. (5.8 × 30.5 m.) reverberatory furnace, which had been erected to smelt the Cananea blast-furnace flue-dust, did not do satisfactory work on such coals as could be obtained, and was changed over to coal-dust firing. After many difficulties, a fair measure of success was obtained and preparations for a larger plant were in progress when the duty on fuel-oil, hitherto prohibitive, was modified by the government of Mexico to such an extent that it became evident that it would be by far the most advantageous fuel.

In preparing the furnace for burning oil, the fire-box was removed, and the end bricked up just behind the bridge-wall, this rear wall being pierced for four burner-nozzles.

The oil employed was a fuel-oil from Oklahoma, which had undergone a slight distillation to remove the gas and light paraffins. It weighs about 7.4 lb. per gallon at a temperature of 17 deg. C., and contains about 19,500 B.t.u. per lb. avoirdupois.

As is invariably the case when expert assistance is wanting, considerable trouble was experienced with the burners. The Hammel burner was used at first, as being widely and successfully employed in this country on locomotives and stationary boilers. While economical and efficient, both this — and, sub-

the place of constant references to the individual authors. "Oil Burners for Reverberatory Furnaces," by Charles F. Shelby, Supt. Reduction Division, Cananea Consolidated Copper Co., *Engineering and Mining Journal*, Jan. 1, 1910. "Oil-fired Reverberatory Furnaces," by R. L. Herrick, *Mines and Minerals*, Jan., 1910. "Experiments in Reverberatory Practice," L. D. Ricketts, General Manager Cananea Consolidated Copper Co., *Inst. Mng. & Metallurgy*, Nov., 1909. Private communications from E. H. Hamilton, Supt., and Cyrus Robinson, Constructing Engineer, Humboldt Smelter, Arizona, and others.

sequently, the Kittle burner — gave an intense heat at the firing end, but left the more distant extremity of the furnace too cold for proper work. As we know from experience that good coal can maintain a suitable heat over the entire extent of a hearth even longer than the one on trial, it was argued that oil should effect the same result even more perfectly, and that, doubtless, combustion was too instantaneous and local. As pointed out by Mr. Shelby, the design of all burners which are devised for atomizing oil under boilers is such as to obtain instantaneously the finest possible atomization of the liquid, in order to get quick combustion and spread the flame. With the long reverberatory hearth, however, almost a contrary effect was desired: namely, to impart great velocity to the atomized particles, and carry them along as far as possible before they should flash into combustion.

With this object in view, Mr. Shelby finally constructed a burner (not patented) which was found to accomplish this end successfully, and to give results that must be considered as highly satisfactory. As the burner is really the critical feature in this kind of smelting, I believe it to be worth while to introduce Mr. Shelby's cuts and explanations.

"The burner is mounted on a pedestal and with a universal joint, in order that the direction of the flame may be controlled. The oil enters through the pipe marked 'oil-supply,' passes into the annular chamber shown, and is discharged therefrom through an orifice $\frac{7}{8}$ in. (0.022 m.) in diameter; this is done with a pressure pump, at about 40 lb. pressure. The steam, at about 125 lb. pressure, comes in through another pipe which fits within the oil chamber, thereby partially heating the oil before it is discharged into the furnace. The steam passes out through an annular opening, the outside diameter of which is $\frac{2}{3}\frac{5}{2}$ in. (0.020), the internal diameter being $\frac{3}{4}$ in. (0.018 m.), thus giving an aperture $\frac{1}{6}\frac{1}{4}$ in. (0.0004 m.) wide, and comes in contact with the film of oil, forcing it into the furnace with great velocity, and atomizing it thoroughly at the same time.

"There is a steel pin or piston that forms the inner portion of this steam circle, and which is movable by means of a hand-wheel and screw-threads, so that the operator can at any time enlarge this steam opening to its full size of $\frac{2}{3}\frac{5}{2}$ in., the increased volume of steam freeing the steam compartment of any scale or

sediment from the pipe, or from other places in which it may have collected and tended to stop up the permanent steam opening. While doing this, he automatically admits a large volume of steam into the oil-chamber, also blowing this clean in the same operation.

"The general design of the burner and its details are shown in the accompanying illustrations. There is also shown a modified form of nozzle that gives practically the same results, except that it has a tendency to throw the flame still farther into the furnace. The steam aperture in this case has been reduced to $\frac{7}{32}$ in. (0.0055

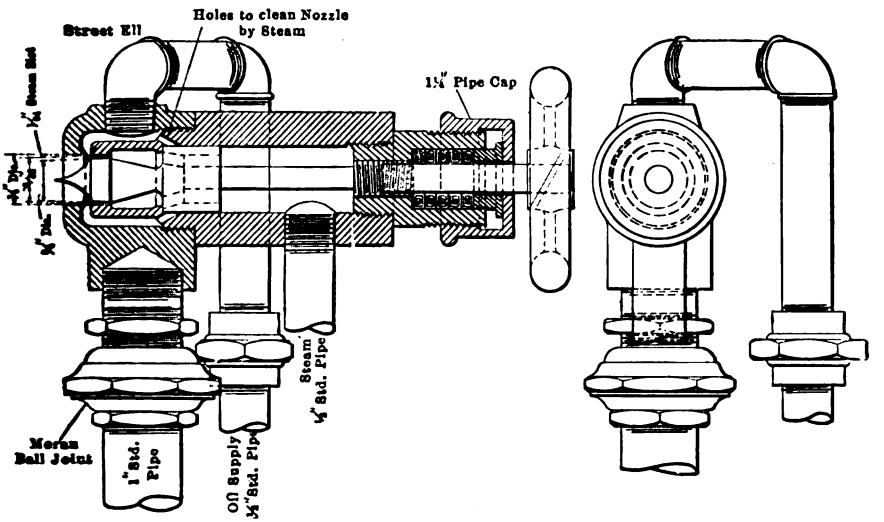


FIG. 73. — Shelby oil-burner

m.) in diameter, in place of the annular opening previously described, both having approximately the same area. At Cananea the modified form of nozzle has given the better results, possibly in view of the finer machine-work that is called for in connection with the other design. Unless the greatest care is exercised in its manufacture, the annular steam aperture will not be true, and the flame will then be exaggerated considerably on one side.

"The dimensions of this burner were arrived at after a careful study of the quantity of steam that is required to atomize properly a given quantity of oil; and these burners, when handling 50 to 60 bbl. (the bbl. contains 42 gallons, or 159 liters) of oil per 24 hours, will atomize it perfectly with 0.3 to 0.4 lb. of steam

per lb. of oil. (One pound = 0.4536 kg.) Or, figuring exactly 0.35 lb. steam per lb. of oil, and an evaporating efficiency of the oil in the boilers of 14 lb. water per lb. oil, these burners then employ about 2½ per cent of the oil for their own operation.

"With the burner here described, the maximum quantity of steam that can pass through the same is beyond the control of the operator, and is only changeable with the steam pressure, which seldom varies to any considerable extent in well operated plants. At Cananea, the steam valves to these apertures are left wide open all the time, and the quantity of steam consumed is considered beyond the province of the furnace-man; the oil being held at a constant quantity, the results are then proportional."

Mr. Ricketts in his summary of results at Cananea, presented lately to the Institute of Mining and Metallurgy,[1] supplements the preceding statement with many valuable details. He refers particularly to the burning-out of the furnace arch which, during the earlier period of these oil-burning experiments, occurred to a painful extent. This was found to be due to inexperience in the management of the burners, especially in permitting the flame to impinge upon the roof, which was constructed of silica brick only 12 in. (0.305 m.) in length.

With the skill gained by practice, with the improved burner, and with a new arch of 20 in. (0.508 m.) silica brick, he is confident that this trouble can be mitigated.[1]

The waste gases are used to heat three 300-H.P. Stirling boilers set in parallel. The temperature of the gases in the flue shortly after they have left the furnace is 1375 deg. C.; at a bend in the flue some 40 ft. (12.19 m.) further along, 1230 deg. C.; at the point where they enter the boilers, 1150 deg. C. The gases as they escape from the boilers average about 400 deg. C.

These figures denote that the gases escape from the furnace

[1] *Opus cit.*

[1] While experience in Montana has shown that it is perfectly feasible to make a silica brick 20 in. long that shall withstand satisfactorily the severe conditions of a reverberatory roof, experience at various other smelters has shown that unless these brick are manufactured in a manner which fits them for this peculiar purpose, they are likely to spall and go to pieces.

while they are still capable of doing effective work in the smelting process, and that there is an excessive loss of heat in the flue between furnace and boilers. Improvements to remedy these faults are already planned; yet in spite of all this waste, exhaustive tests show that about 50 per cent of the original heat-value of the oil is recovered in the boilers.

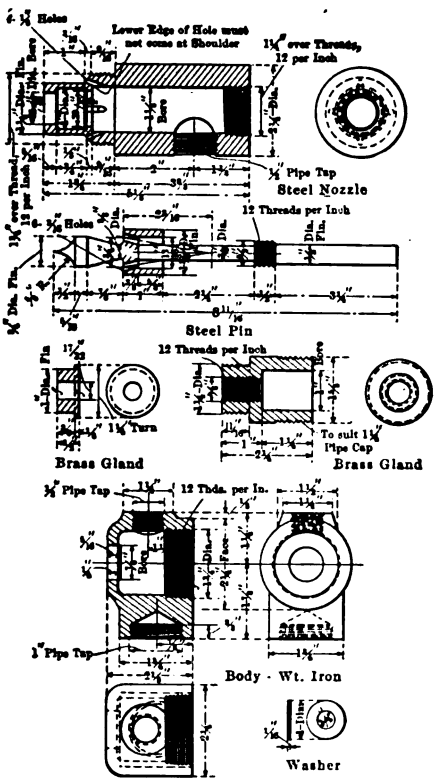


FIG. 74. — Shelby oil-burner — details

Systematic tests of new practice, covering long periods of time and emanating from an unquestionable source, are of such inestimable value to the profession that I feel constrained to devote more space to the details of Mr. Ricketts' statement than I can accord to most subjects.

The oil arrives at the smelter in tank-cars and flows by gravity

to three storage tanks, each of 10,000 bbl. capacity,[1] whence it is piped to smaller tanks at the smelting furnace, power-house, etc. At the reverberatory furnace a 24-hour supply of oil is stored in two tanks, each having a capacity of 110 bbl., and carefully gauged and marked so that the hourly consumption of the burners can be followed with some degree of accuracy.

The four burners enter the rear wall close above the hearth through holes about 12 in. (0.305 m.) in diameter. They converge slightly toward the central longitudinal axis of the furnace, and at an angle which must be adjusted with great precision so that the jets shall not in any way impinge upon the side-walls nor yet converge so strongly that they meet while still retaining too much of their initial velocity; otherwise, confused, whirling eddies of burning gases rush upward against the arch with disastrous effect. Under careful management a roof should last six months before coming to any serious patching.

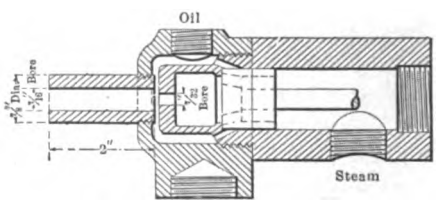


FIG. 75. — Shelby oil-burner — Modified type

During the first six months of 1909 the reverberatory ran 165.7 days, as follows:

INTO FURNACE		
flue-dust	29,802	tons
roasted concentrates	5,484	"
silicious ores	2,300	"
fluxes	3,129	"
	40,715	"

OUT OF FURNACE		
matte and foul slag	10,120	tons
clean slag	25,658	"
volatilized (by difference)	4,937	"
	40,715	"

[1] One barrel contains 42 gallons, or 159 lit.

The percentage of copper in the charge was 7.08, and of sulphur, 9.77, the ratio of sulphur to copper being 1.38 to one. The percentage of copper in the matte was 30.57, and the recovery of copper in the matte, 96.04. The average daily duty of the furnace was 245 tons. (One ton = 0.907 met. t.)

OIL CONSUMPTION

barrels of oil consumed	39,417
barrels of oil credited to steam production	17,471
net gallons of water evaporated	9,883,493
barrels of oil charged to smelting	21,946
gross bbl. oil consumed per dry ton charge	0.968
bbls. oil per dry ton smelted credited to steam	0.430
net bbls. oil per dry ton charged to smelting	0 538
average net H.P.	664.3

The actual water evaporated in the boilers per pound of oil burned in the reverberatory was 6.74 lb., whereas the pound of oil burned under the power-house boilers evaporated 13.96 lb. water. Taking account of the temperature of feed-water, and converting into evaporation from and at 100 deg. C., we have the following figures: evaporation (by waste gases) per pound of oil burned in reverberatory, 7.45; evaporation per pound of oil burned under power-house boilers, 16.08.

The average slag analysis during this period of six months was

SiO_2	40.10	per cent
Al_2O_3	11.20	" "
FeO	34.80	" "
CaO	8.50	" "
ZnO	2.30	" "
MgO	0.40	" "
MnO	0.30	" "
S	0.70	" "
Cu	0.45	" "
	98.75	" "

Mr. Ricketts also furnishes the total cost of smelting per ton of dry ore. This statement does not include the cost of roasting the concentrates, and the weights used are the weights of the material after it has been roasted. The shrinkage in weight in roasting dry Cananea concentrates is about 15 per cent.

COST PER TON ORE SMELTED, FIRST SIX MONTHS OF 1909

labor, operation	$0.24
labor, repairs	0.16
supplies	0.22
fuel-oil	1.73
power	0.02
miscellaneous	0.08
total direct costs	2.45
cost of fluxes	0.08
	2.53
credit for power supplied	0.77
net cost	1.76[1]

During one month, in which a much larger proportion of the easily-fusible roasted concentrates was used, the net cost per dry ton of ore smelted was $1.29, after $0.67 had been deducted from the gross cost on account of steam credit.

It will be noted, therefore, that the same furnace which would smelt about 100 tons per 24 hours on (unsuitable) coal, and 180 tons with pulverized coal (during the short periods when the flues were clear of ash), has risen to an average of 245 tons, on a consumption of about 38 gallons of oil per ton of ore (143.4 lit.), and is recovering from the waste gases, in the shape of steam power, 46.33 per cent as much effect as though the oil had been used direct for firing boilers.

The amount of labor is reduced materially by the use of a fuel which is fed automatically, requires no leveling or stirring, and burns without any solid residuum. Where coal is employed, there must be labor expended to bring it to the hoppers above the furnace; a certain amount of work is required to drop the coal at regular intervals into the fire-box, as well as to level and stir the fire; long and exhausting labor is needed to keep the grate in proper condition; and, finally, a sluicing and jigging system is required to wash away the droppings from the grates and reclaim their valuable portion. All this is eliminated by the use of a liquid fuel which flashes almost instantaneously into gas, and which is consumed without ash.

R. L. Herrick studied the work at Cananea toward the close of 1909, and states that the entire force employed at the furnace,

[1] This statement evidently omits certain general items which form a legitimate part of the actual costs of smelting.

when matters were in normal condition, consisted of one American foreman at $3 per day and three Mexicans at $1 per day, working on 10-hour shifts. To this must be added the extra labor required for repairs, which, according to the six months' figures given by the general manager, amounts to 75 per cent as much as the regular labor. This includes, however, the extensive repairs due to inexperience in the management of the burners, and promises to become diminished to a considerable extent.

In 1906 the Arizona Smelting Co. erected a custom smelter at Humboldt, Arizona, the most interesting feature of which consisted of two large reverberatories built especially for burning fuel-oil. It ran successfully until the drop in the price of copper occasioned a temporary stoppage; it is now again in operation.

A custom smelter in a silicious gold-silver — rather than copper — district can offer no such uniform and striking results as those of the preceding illustration. On the other hand, I find it a peculiarly instructive situation from the fact that its technical superintendent, E. H. Hamilton, came direct from the coal-burning furnaces of the Washoe smelter, with its uniform and fusible charge, to the oil-fired reverberatories of Humboldt, with its variable and constantly silicious mixture.

The change from coal to oil as a fuel is alone sufficiently radical to introduce a long series of complications and difficulties until the new fuel, new conditions, and new workmen have adjusted themselves mutually; but when to all this is added a novel, difficult, and variable charge, the new fuel can scarcely be expected to make a reasonable showing, unless it possesses such decided advantages that it can overcome this heavy handicap and still have margin enough remaining to hold its own. That it was able to accomplish this speedily and without excessive experimental work speaks well both for the fuel and the management.[1]

The Humboldt reverberatories have neither fire-box nor

[1] My principal materials for the following statements are derived from a graduation thesis by Junjiro Kobuse, of the Royal University of Tokio, who spent several weeks studying the Humboldt works; from private letters from Cyrus Robinson, constructing engineer of the Arizona Smelting Co., and from J. L. Elliot, president of the same; also from a short paper by E. H. Hamilton, general superintendent of the company, published in the *Engineering and Mining Journal* of May 11, 1907, p. 901.

bridge proper. The inside length of the hearth is 98 ft. (one foot = 0.3048 m.). Its width is 19.08 ft.[1]

The openings for the burners consist of five slightly arched apertures in the rear wall 1 × 1.4 ft. in size, and spaced 3.16 ft. from center to center. Small checker-holes for the admission of air are provided above these larger apertures.

After some experimenting, Kirk's burners were used, being hung on universal connections so that they could be pointed at any angle. The oil system is arranged so that there is a continuous and converting flow from an electrically-operated plunger-pump, under a pressure of 80 lb. per sq. in., the excess oil returning to the pump with the suction. The steam is taken from the waste-heat boilers, and approximates seven per cent of their duty. Connected in multiple with the two furnaces are two batteries of Stirling water-tube boilers, having, in total, 13,000 sq. ft. (1208 sq. m.) of heating surface, giving about 1300 b.h.p. The flue connections are so arranged that any one boiler may be cut out and the waste gases of the furnace sent to the other boilers; or one reverberatory may be run on any two boilers without interfering with the other two. All of the boilers are equipped with oil-burners, thus permitting steam to be kept up independently of the furnaces. These boilers are the only ones upon the plant, and furnish all the steam required for operating the smelter, a 400-ton concentrator, and a mine some 10 miles distant.

Owing to the reasons already mentioned, the character of the charge is extremely variable. It is, therefore, impossible to offer exact figures of average results. It may be said, in the main, that the smelting operation at Humboldt has too much silica, too much sulphur, too little iron, and too little copper. These are all unpleasant factors, and the ability to deal with them at all, and still maintain a margin of profit on the comparatively moderate tariff established, is due in large part to the quick heating and high temperature rendered possible by the nature of the fuel.

Two points are especially noticeable in studying the daily behavior of the furnace on its variable mixture.[2]

[1] Plates and description of these furnaces may be found in "Principles of Copper Smelting," p. 210.

[2] I have been permitted to study these results in some detail, but can only offer my own general impressions derived therefrom.

The first striking feature of the process is the unusual proportion of sulphur removed in the fusion. This result is the more surprising because there is an unusual proportion of unroasted ore in the charge which does not contain enough sulphur to make it worth while to submit it to the roasting process, and yet is in a peculiarly unfavorable condition for any removal of sulphur during its fusion. To be sure, it will lose such proportion of its sulphur as is dissociated by heat alone according to the familiar formula, FeS_2 + heat = FeS + S (the sulphur vapor burning to SO_2 gas as soon as it comes in contact with air);[1] but this is but small compared with the driving out of sulphur which occurs from the reaction of oxides upon sulphides, as in smelting ordinary calcines. Consequently, with a given ratio of sulphur to copper, we should expect the proportional elimination of sulphur during the fusion at Humboldt to be lower than at smelters where the charge consists mainly of roasted sulphides, and thus reaps the benefit of the admirable reactions between oxides and sulphides.

At the great Montana reverberatories, running on roasted sulphides, it is necessary — in order to produce a 40 per cent matte — that the ratio between sulphur and copper in the charge should not be above one to one. At Humboldt, however, a 40 per cent matte is produced from a charge containing 10, and even 12, per cent sulphur to five per cent copper — which is two sulphur to one copper. This is attributed to the greater oxidizing effect accompanying the comparatively perfect combustion of oil.

The second noticeable feature at Humboldt is the high silica content of the slags, reaching 45 per cent at times, and while employing ores as coarse as two and two and one-half inches (0.051 to 0.063 m.). Of course, the capacity of the furnace (which was said to approximate 300 tons, 272 m. t., per day on a favorable mixture) was reduced very materially; yet the management believed that it would have been commercially impossible to smelt these silicious ores at all with coal.[2]

[1] This formula, though not scientifically exact, is quite sufficient for ordinary purposes. See page 44 for a more accurate study of this important reaction.

[2] The average silica content of the reverberatory slag at the Humboldt smelter, as reported at the New York office, during the first six months of 1907, was: 45.6, 43.0, 44.1, 42.8, 43.1, 43.8, 43.7 per cent. The iron averaged about 23 per cent (metallic). As might be supposed, with acid, well-melted

The Peyton Chemical Company of California constructed, in 1907, a regenerative oil-burning reverberatory for smelting copper ores. The reason for adopting the regenerative system arose from the fact that their original oil-burning furnace heated in an irregular and unsatisfactory manner, giving a high temperature at the combustion-end of the hearth and a low one at the opposite extremity; thus the slag chilled at the skim-door, the hearth area was not fully utilized, and the capacity was small.

While I believe that these troubles could have been overcome by the use of suitable burners, it is instructive to study the application of the regenerative principle to the conditions of the copper smelter, and it is quite possible that the experience of the Peyton Chemical Co. may prove an aid to some isolated reverberatory plant which, depending upon poor coal, lignite, wood, or taquia for its fuel, finds its heating capacity a little below the desired standard. The reason why I consider the following experience of such especial value to our peculiar conditions is that it calls for no checker-work, which is the worst feature of regenerative systems in connection with copper smelting. [1]

Owing to local conditions, regenerators underneath the hearth were impracticable. Ordinary checker-work is unsuitable on account of rapidly forming accretions of flue-dust, while the Whitworth stove was not favored, owing to the fact that it would be useless should the experiment prove unsatisfactory.

Mr. Leas adopted the following plan, which answered well in practice, and seems particularly suited to the conditions which the copper smelter is often called upon to encounter.

The furnace was to be small — only 20 × 40 ft. (6.1 × 12.2 m.) — and was built practically of equal width for its entire length, thus approximating a parallelogram. The revert was

slag, low in iron, the copper in the slag was low, averaging 0.27 per cent for the six months. The matte was also rather low most of the time, copper ores being scarce, and the smelter depending mainly upon gold and silver. I think it averaged about 30 per cent copper during those six months, and thus came within the rule that the slag should not contain more copper than may be indicated by moving the decimal point of the matte-assay two places to the left.

[1] I make free use of a paper by Fred A. Leas, entitled "Regenerative Reverberatory Copper Furnace," *Engineering and Mining Journal*, Nov. 7, 1908.

constructed in the usual manner, and the side-walls were built to an even height all around, so that the roof was horizontal throughout its entire length. The usual flue connections were provided at each end, and led each to its respective downtake, which entered a flue at right angles to the long axis of the furnace, and 150 ft. (45.7 m.) in length. There were, thus, two parallel flues, separated from each other by the length of the furnace. At the outer end these flues were brought together by turning toward each other at right angles, and from this point of junction a single flue led to the steel stack. Where the flues joined, a three-way valve was placed in such a manner that one flue would discharge its gases into the stack, while the other would be open to the atmosphere for admission to the hearth of the preheated air necessary for combustion.

The oil at the six burners is atomized by air from a Connersville high-pressure blower at seven to ten lb. per sq. in., each burner requiring about 50 cu. ft. free air per minute.

The regenerative system proved satisfactory, the increase in capacity over the former oil-practice being fully fifty per cent. Under the older conditions, the furnace smelted 60 to 65 tons (54.4 to 59 m. t.) per 24 hours, while the improved plan brought it to an average of 90 tons (81.6 m. t.).

The reasons for so great an increase in capacity are obvious. The entire hearth area is now utilized all of the time, and the temperature of the entering air is raised to a point ranging from 400 deg. to 1000 deg. C. — depending upon the frequency with which the current is reversed. It was found advisable to reverse the current once every 60 to 90 minutes, dependent upon the fusibility of the charge. It was not found beneficial to change fires as often as in open-hearth practice, for the reason that the area in the flues was so large that the heat was not taken as rapidly by radiation to the air traveling back to the furnace, and also to the outside atmosphere, as it was taken up in the cold flue. This shows that the regenerative effect could have been accomplished in a much shorter flue; but, in this case, the large flue space was desired in order to effect a satisfactory settling of the dust from the finely pulverized charge.

As additional capacity was needed, a second and larger furnace was built, having inside hearth dimensions of 17 × 60 ft. (5.18 × 18.3). While the same general plan was followed of con-

necting the furnace with the regenerative flues, the latter were made to occupy less size — as well as to settle the dust more efficiently — by changing their direction six times at right angles during their course to the stack.

This furnace, with an available hearth area of 1020 sq. ft. (94.8 sq. m.), has smelted under ordinarily favorable conditions 176 tons (159.6 m. t.) per day, the charge consisting of 56 per cent hot calcines, 25 per cent cold calcines, and 15 per cent siliceous copper ore. This gives a capacity per sq. ft. hearth area of 345 lb. (1683 kg. per sq. m.) per 24 hours. The original oil-burning furnace smelted about 200 lb. and, after being made regenerative, about 300 lb. per sq. ft. Mr. Leas believes that, if running with such a charge as is customary at Butte (hot calcines, with a 20 per cent matte-fall), the larger furnace could smelt 200 tons (181.4 m. t.) per day.

The subject seems to me well worthy the consideration of metallurgists who labor under conditions to which I have already referred.

The most important points to bear in mind are: that it was the employment of simple, long flues, instead of checker-work or stoves, that robbed the regenerative system of its terrors for the copper smelter; and that the construction was made cheap and simple by the double purpose to which these flues were put. Provision could be made for rapid removal of the flue-dust without checking sensibly the duty of the furnace.

As the temperature of the gases after they go to waste is reported as 500 deg. C., an installation of boilers, or preheaters for boilers, would be well worth while where fuel is expensive.

The use of wood as a fuel in reverberatory smelting is of great importance in new districts. As it involves certain peculiarities of construction and management, it can be treated more clearly and definitely in a chapter by itself.

We have studied, up to this point, the construction and management of the reverberatory furnace, and have learned how we may best produce and control the high temperature needful for the melting of ores, and how we may confine and remove the unusual quantity of molten material with which we have to deal.

We have studied in another chapter the behavior of our ordinary substances at elevated temperatures, but there remains yet to consider how these substances react upon each other, during their fusion upon the reverberatory hearth, in such a manner that they yield the objects of our efforts: namely, a clean, fusible slag, and a suitable matte containing the copper (incidentally, also, the gold and silver) which was originally disseminated throughout the particles of ore of which the charge was composed.[1]

As is the case in all furnace work, the class of reactions which occur between the constituents of the ore, when exposed to a high temperature, depends largely upon the kind of atmosphere that prevails within the furnace. While no one doubts that the true pyrite furnace has an intensely oxidizing atmosphere at its zone of combustion, nor denies that the true coke-burning blast-furnace possesses a strongly reducing atmosphere, there are those who object to the customary characterization of "neutral" for the atmosphere of the reverberatory smelter. This point seems to be scarcely worthy of controversy. Whatever we may choose to call its atmosphere, its results approximate closely to those yielded by the fusion of a similar mixture in crucibles in a neutral atmosphere, as may be substantiated with ease at any smelter. As the other two atmospheres are already appropriated, and as it seems convenient to provide every process with some kind of an atmosphere for purposes of description and discussion, I shall continue to assume that — on the average — the copper-smelting reverberatory has a neutral atmosphere.

This enables us to observe the reactions of the different constituents of the charge upon each other more clearly than in processes where the natural affinities are constrained and obscured by the influence of the atmosphere which surrounds them.

In reverberatory smelting we deal for the most part with mixtures consisting of the higher iron oxides (Fe_3O_4, Fe_2O_3), of the copper oxides (Cu_2O, CuO), of silica, of the earths (classing alumina among the earths), and of a certain amount of unoxidized sulphides. There are also present various sulphates of

[1] It is not the object of this book to go exhaustively into this side of the subject. I have studied this at length in a former work, and refer to "Principles of Copper Smelting," Chapter VII, for a detailed discussion of the theory of the reverberatory process.

copper and iron, as well as of the earths. As usual, we desire to have the silica, the earths, and a considerable proportion of the iron unite to form a slag, while the copper oxides shall exchange their oxygen for sulphur and become Cu_2S, and a proper proportion of the iron shall join it as FeS, thus forming a matte of the desired grade.

For purposes of simplification, we may at once eliminate such bases and acids as will enter into chemical combination with each other without complications and in the condition in which they already exist. Thus, we may at once cancel all of the earths against a certain portion of the silica, and thus relieve our problem of CaO, MgO, BaO, and Al_2O_3, and a specific portion of the silica, leaving for our study only the iron and copper oxides, sulphides, and sulphates, together with the remainder of the silica.

Considering carefully the mixture now before us, it becomes evident that we have need of a reducing agent for two distinct purposes: first, for reducing the copper oxide so that it may not combine with silica and form slag; second, for reducing the higher iron oxides to FeO, which is the only condition in which iron can combine with silica to form a good slag. According to our theory, we can ask no such reducing aid from the atmosphere of the hearth, and are thrown back upon the ore-mixture itself. As every one knows, the sulphur still remaining in the imperfectly roasted (or added, raw) ore plays this reducing rôle, and renders possible what would otherwise be an impossibility: namely, the smelting of roasted sulphide copper ores in the reverberatory furnace, without slagging the valuable metal.

In the coke blast-furnace we have no need of this reducing influence of the sulphur — although even there it plays the same part in a subordinate form — for the powerful action of the CO is eminently calculated to reduce both the copper and the iron oxides; hence it arises that the most striking feature of the reverberatory process of smelting roasted copper ores is the fact that sulphur acts as the sole reducing agent for the oxidized compounds of both copper and iron (the latter being reduced only as far as FeO).

To aid the imagination and memory, it is convenient to express these characteristic reactions by means of chemical formulæ, of which a great variety may be conceived: for instance,

$$Cu_2S + 2CuO = 4Cu + SO_2$$
$$Cu_2S + 2Cu_2O = 6Cu + SO_2$$
$$Cu_2S + 3CuO = 3Cu + Cu_2O + SO_2$$
$$Cu_2S + 6CuO = 4Cu_2O + SO_2$$
$$Cu_2S + CuSO_4 = 3Cu + 2SO_2$$
$$Cu_2S + 4CuSO_4 = 5CuO + 5SO_2$$

The metallic copper thus formed by the reactions between the sulphide and the oxide compounds in the roasted ore has a great affinity for sulphur, and takes this element from the FeS, as follows:

$$2Cu + 2FeS = Cu_2S + FeS + Fe$$

the Fe thus formed oxidizing at once to FeO and combining with silica.

At a higher temperature the following reaction takes place between copper oxides, iron sulphide, and silica:

$$FeS + Cu_2O + xSiO_2 = Cu_2S + (FeO + xSiO_2)$$
$$4FeS + 6CuO + xSiO_2 = 3Cu_2S + (4FeO + xSiO_2) + SO_2$$

The higher oxides of iron are reduced to FeO as follows:

$$FeS + 3Fe_2O_3 + xSiO_2 = (7FeO + xSiO_2) + SO_2$$

It will be seen, therefore, that a certain proportion of the sulphur in the roasted ore acts as a protector for the copper by reducing the oxides of that metal — itself burning to SO_2 — while another portion of the sulphur combines with the reduced copper to form Cu_2S. Still another fraction of the sulphur burns to SO_2 in reducing to FeO the higher oxides of iron, and it is by these reactions, which cannot occur freely in a reducing atmosphere, that so considerable a proportion of the sulphur is removed when smelting in the reverberatory furnace a mixture of oxides and sulphides.

This series of reactions is sometimes carefully planned, and carried out in such perfection that a rich copper matte, free from iron, is converted entirely into metallic copper by a simple melting-down fusion, without any aid from the oxygen of the atmosphere.

In the Nicholls and James direct process of refining, as practised since 1891 at the works of the Cape Copper Co., in Wales, two parts of thoroughly roasted white metal (CuO) are melted with one part of raw white metal (Cu_2S) with the production of metallic copper and SO_2 gas: thus, $Cu_2S + 2CuO = 4Cu + SO_2$.

Gas-fired furnaces, with regenerative appliances, are used very little for copper smelting in North America; in fact, the only large plant using producer-gas is now planning to procure better coal, and give up gas-firing completely. It is only just to say that much of the dissatisfaction with the use of producer-gas arises from the poor quality of the coal employed; this coal, at the Boston and Montana smelter, contains 20 to 35 per cent ash, and is difficult to burn in the producers.

Constant difficulty is experienced in the choking and slagging-away of the checker-work from the ore-dust and, while the heating power of the furnaces is good, and their capacity large when in good condition, the delays, repairs, and renewals are very heavy.

CHAPTER XI

REVERBERATORY SMELTING WITH WOOD

THE twenty years of time between 1868 and 1888 witnessed the inception, rise, decline, and practical extinction of the wood-burning reverberatory smelter at any of the great copper plants of the United States. The advent of the railroad and the disappearance of the forest influenced the metallurgist to rebuild his furnaces for the use of the more concentrated and manageable fuel.

My own early experience covered much of this wood-burning period and, until 1884, I looked upon coal as a novel, and somewhat experimental, fuel to employ in a reverberatory smelting furnace.

I know of no author writing upon the metallurgy of copper who has devoted any considerable portion of a text-book to the serious study of the wood-burning reverberatory from its practical side; but the numerous requests for advice, information, plans, and general data upon this subject which have reached me during the past ten years indicate that the use of wood as a fuel in this class of work is a matter of interest and importance to a large number of metallurgists in different parts of the world, and I intend, in this chapter, to give such information as my own experience and observation — aided by the scanty literature of the subject — enable me to offer, believing that it will be especially useful in Mexico, where there are many known — and, doubtless, still more unknown — districts situated within the great wood-belt of the Sierra Madre that are debarred from railroads and custom-smelters, and that would gladly turn their richer ores and concentrates into a high-grade shipping product by the use of wood as a fuel, provided they could obtain reliable information as to the manner of employing it and the results which they might expect therefrom.[1]

[1] In order not to waste time in repeating information already given, I must ask serious readers to use this chapter in connection with the section

I may say, at the outset, that there is no radical difference between the coal-burning and the wood-burning reverberatory process; merely one of degree. There is much the same difference, to use a homely illustration, that exists between a typical athlete and a man of very ordinary powers; yet the aid of the ordinary man may be exceedingly welcome in cases where that of the athlete is not available.

As the sole cause of difference between the two types of furnace lies in the fuel, and as we are already familiar with the qualities of coal and with its application to the smelting of copper ores in the reverberatory furnace, it is plain that, before we can institute any instructive comparison between the two fuels, we must learn a few essential facts regarding the composition and combustion of wood.

Wood is composed essentially of organic tissue and a very small proportion of inorganic matter. In its natural condition it contains a large proportion of water, which it loses at a temperature considerably below that at which the decomposition of the organic tissue begins.

Apart from small quantities of tannin, resin, etc., the only important combustible portion of the wood is cellulose, and this material is believed to have the same general composition for all varieties of wood. Despite this belief, the analyses of cellulose, as given by distinguished chemists, differ materially. This fact, however, has no bearing upon the present issue, and it is sufficient to recognize that the three constituents of cellulose exist, approximately, in the following proportions:

carbon	48	per cent
oxygen	46	" "
hydrogen	6	" "
	100	" "

The amount of ash produced by the complete combustion of wood is so small that it is of no practical importance — the more so that it may be regarded as basic in character, having a com-

on "Reverberatory Furnaces" in this book, and Chapter VII on the same topic in "Principles of Copper Smelting." The present chapter is not complete in itself, owing to the omission of much that has already been discussed at length in connection with coal-burning reverberatories.

position that may be represented in a general way by the following composite table which I have constructed from a large number of analyses made for the French and German departments of forestry:[1]

COMPOSITE ANALYSIS OF THE ASH OF BARK-COVERED WOOD DRIED AT 100 DEG. C.

SiO_2	8	per cent
CaO and MgO	58	" "
FeO and MnO	8	" "
K_2O and Na_2O	18	" "
	92	" "

The remaining eight per cent consists of phosphoric, sulphuric, and carbonic acids, a small amount of chlorine, and minute quantities of various substances.

The attention of the practical smelter should be directed mainly upon the amount of water which his wood contains, and which is of far more importance to him than any other subject pertaining to the composition of his fuel. While much of the water contained in the wood is an essential part of its make-up, it may be regarded as being held mechanically by the porous cellulose rather than forming any integral part of the same; thus, to a certain degree, resembling the manner in which water is held by a sponge.

During the process of combustion all of this water must be evaporated by the remaining constituents of the wood, and any one familiar with the great amount of heat required to convert water into a vapor will appreciate at once the immense drag upon the useful effect of the fuel which must result from applying so large a proportion of its activity to the mere expulsion of a harmful constituent.

The reverberatory furnace is too costly and too highly-organized an apparatus to be used as a dry-kiln. We may, when we find it worth while, employ its waste gases to evaporate water

[1] Both the quantity and the composition of the ash vary astonishingly according to the character of soil and climate, and according to whether the sample analyzed contained its proportionate amount of bark, the latter substance yielding several times as much ash as the naked wood. In the analysis here given, all the samples have contained bark.

for power, because we are through with these gases, and it is always good practice to work up a by-product when it can be done profitably. But to begin at the other end and diminish our all too scanty heat supply by employing it for the profitless evaporation of water is simply suicidal. It is impossible to speak too strongly on this matter, and the economy — and often the existence — of an enterprise depends upon the appreciation and application of this especial point. Metallurgical writers on copper have been urged for some years to give detailed information regarding the employment of wood in copper smelting, and to point out the especial peculiarities and precautions attending its use. *This* is the especial peculiarity and precaution, beside which most other details are insignificant.

The troubles arising from this cause are doubled when the new smelting enterprise is started without an ample supply of properly seasoned wood. A cold furnace, untested bottom, experimental ore-mixtures, ignorant workmen, and often a manager inexperienced in this particular branch of smelting, form a handicap that is quite sufficient even when all other conditions are favorable; but if to this is added the overpowering burden of unsatisfactory fuel, the chances are that wood-smelting will be pronounced a failure even under circumstances particularly adapted to its employment.

It is difficult, especially for a young superintendent, to induce his company to invest a considerable sum of money in providing, in advance, a liberal supply of seasoned wood in addition to the heavy outlay involved in the construction of a new furnace plant. It is doubly difficult to do so if, when first called upon for estimates of cost for the new departure, he makes the mistake of limiting himself to the actual cost of the plant.

I learned early, from bitter experience, that a permanent investment in seasoning cord-wood should be included as a part of the capital required to start the operation, and that the way to ensure this capital was to include it in the original estimate as a part of the sum required to erect the furnace. At the inception of the enterprise, and when the question of raising the capital necessary to start a new smelter is interesting the directors of the company, a few thousand dollars more or less seems a matter of small moment to them; but after the money is provided, and they are beginning to count on speedy returns, a second call for a

floating capital to invest in wood comes as a most unpleasant surprise, and seldom meets with a cheerful response.

The time of making the original estimate is the time to begin educating your directors to the fact that there are three main items of cost in adding a wood-burning reverberatory to the equipment of the mine.

(*a*) The furnace itself, with a tolerable supply of fire-brick and refractory materials, to tide over the first few months.

(*b*) The buildings, track, tools, and accessories (often, a roasting-plant).

(*c*) A several months' supply of cord-wood.

This latter item must always be kept good by renewals as fast as it is used up, and resembles the floating balance, of perhaps $100, which a small depositor is required to maintain if he wishes to keep an account at a bank. He still owns the hundred dollars and may withdraw it whenever he wishes to close his bank account; but so long as he desires to maintain his banking privileges, he must keep it good.

If the company is unable or unwilling to accede to this conservative arrangement, the next best plan is to explain fully, in writing, the increased cost and imperfections that will result from this omission, and to secure at least one or two hundred cords of seasoned wood with which to smelt-in the bottom and tide over the period while the men are being educated to the new work. One can then at least start with a sound bottom, and can determine what sort of results may be expected from the employment of proper fuel, and will not decide that the smelting is a failure because green wood fails to produce sufficient heat for profitable work.

Recognizing, then, the fatal disadvantages attending the use of wood containing too much water, it becomes necessary to learn something as to the normal composition of freshly-cut wood, and of the means at our disposition for removing this water to a reasonable extent. We cannot remove it anything like completely, as the remedy would be worse than the disease, the cost of any thorough artificial methods of drying it being far too great to be considered.

The actual proportion of water in wood is extremely variable, differing with the variety of tree, and also being much affected by climate and soil. The impossibility of establishing any average figure for it may be seen from the fact, stated by Percy, that even

in the same tree the young wood contains nearly twice as much water as the trunk.

In climates having an average rainfall, the varieties of wood which might ordinarily be used for fuel, and cut in the spring or summer when the sap is up, will contain from 40 to 50 per cent moisture, running from 8 to 10 per cent less if cut in the autumn or early winter.

In arid climates the proportion of water is decidedly less. Six months of air-drying removes from 40 to 50 per cent of this water, provided the wood is not completely covered with bark. In the latter case, the rate of drying is so retarded as to appear incredible, although the united evidence of various European official forest laboratories forces us to believe it.

This fact explains the exceedingly poor heating qualities of the small round sticks which are so abundant in most deliveries of wood in the Rocky Mountains and Mexico, and emphasizes the importance of minimizing their effect by insisting upon the splitting of every stick beyond a certain moderate diameter. The difference in the rate of drying between a round stick and one that has been split is astonishing. The removal of the bark would exert an even more favorable influence upon the drying, but is manifestly impracticable.

In the dry mountain and plateau regions of Mexico and the western United States, oak, pine, or spruce wood that has been split and dried for six months should not contain more than 12 to 14 per cent water.

The following table of the water-content of slightly-seasoned mountain oak used for making gas is taken from a paper by John Langton, entitled: "The Power Plant of the Moctezuma Copper Company, Nacozari, Sonora, Mexico," *Trans. Am. Inst. Mng. Engrs.*, Vol. XXXIV, p. 748:

WATER-CONTENT OF WOOD USED AT NACOZARI IN 1902

Oct. 22....12.94	Nov. 1....23.42	Nov. 13....21.60
Oct. 23....17.30	Nov. 2....20.97	Nov. 14....23.97
Oct. 24....16.31	Nov. 4....22.92	Nov. 15....25.21
Oct. 25....17.43	Nov. 5....22.19	Nov. 17.... 9.67 (seasoned)
Oct. 27....19.68	Nov. 6....23.25	Nov. 18....22.33
Oct. 28....15.23	Nov. 7....24.24	Nov. 20....18.60
Oct. 29....16.86	Nov. 8....21.48	Nov. 24....16.00
Oct. 31....21.00	Nov. 10....22.08	Average 19.76

The importance of this subject warrants the space which it has occupied.

The wood is seldom accumulated near the smelter in any such quantity as I have indicated. It is more customary to arrange with the contractors to pile it roughly to dry, in places safe from fire, somewhere near where it was cut. A certain advance per cord is made by the smelting company, as no contractor could or would carry so heavy a burden unless paid an exorbitant price per cord. Of course, a fortnight's supply, or more, is always kept piled near the smelter to provide for emergencies, but it is economical to arrange for the main delivery at the fire-box of the furnace. Here, posts and cross-pieces are set up, forming a frame which will hold a certain number of cords. At least two of these frames are provided for each furnace, so that while one is emptying the other is filling.

Assuming that we are to have an ample supply of good split air-dried wood, containing no unreasonable proportion of unsplit sticks, and discarding everything below a certain diameter — perhaps three inches (0.076 m.) — we may next examine into the effect that we may hope to produce in burning it.

Chemical analysis shows us that wood is but a feeble fuel compared with coal, having less than 50 per cent combustible matter whilst fair coal should average above 75 per cent. The wood also is still further handicapped by having more moisture to evaporate, and by being of such a form that it tends to keep too open a fire and to admit more air through the grate than it needs.

It will be shown later how these defects may be minimized by burning the wood in a manner suited to its peculiarities, rather than by following the methods customary for the more concentrated coal.

The scope of this work does not permit detailed thermal calculations, especially of substances which have been studied thoroughly by more competent observers. C. A. Grabill has compiled useful data on this subject,[1] besides making independent calculations, based on Dulong and Petit's law, from which he concludes that the combustion of a good coal may produce a

[1] "Theoretical Notes on Reverberatory Furnaces," *Eng. & Mng. Journal*, Apr. 10, 1910.

maximum theoretical temperature of about 2030 deg. C., while air-dried wood should yield a maximum of 1575 deg. C.

Modifying these theoretical figures by the results of actual work, and using as a check the careful and continuous pyrometric tests of the Washoe smelter on the average temperature of their furnaces above the fire-bridge, when using the excellent Diamond coal described in a previous chapter, we find that 1550 deg. C. is about the maximum effective temperature that we can expect from coal under ordinary conditions. This is 480 deg. C. lower than the theoretical figure, and represents, in my judgment, as high a temperature as I have ever seen in the fire-box of any coal-burning copper reverberatory running on natural draft.[1]

If we assume that 480 deg. C. represents the difference between the theoretical and the actual temperature yielded by coal, and attempt to apply the same correction to wood, we arrive at only $1575 - 480 = 1095$ deg. C. as the maximum temperature which we may hope to maintain, in practice, with air-dried wood.

There is, however, no reason to assume that the same difference exists between the theoretical and the actual combustion temperature of wood as with coal — the gap may be even greater, or it may be far less.

So far as I am aware, we have no reliable, long-continued pyrometric observations concerning the combustion of wood, nor do I feel satisfied at all that 1575 deg C. represents the maximum temperature of its combustion. At any rate I expect to offer conclusive evidence that a temperature considerably above 1095 deg. C. may be maintained continuously by the use of comparatively inferior wood (mountain spruce).

With so many uncertainties in our data, mere calculations are unsatisfactory, and it will be wiser to begin at the other end of the problem and see what results are actually accomplished by the use of wood as a fuel, regardless of thermal calculations or of the fusion-point of slags.

Lack of space forbids my taking up systematically several wood-burning reverberatories (a number of them under my own charge), including furnaces at Alma and Mt. Lincoln in Colorado,

[1] There is no doubt that a higher temperature than this is obtained by the proper use of oil or gas, as evinced by the disastrous effect upon the lining when due to carelessness or inexperience. In the present instance, however, I am confining my remarks to coal as a standard of comparison.

and a considerable number in Butte, Montana. As the greater includes the lesser, I will select for my principal description the plant which for years smelted the most difficult ores, with spruce wood as a fuel, and with results that have not been excelled by any furnaces of the kind with which I am familiar. I refer to the old Boston and Colorado smelter at Black Hawk, Colorado; later removed to Argo; and now, to the sorrow of all of the older generation, only a name.[1]

This smelter was situated at Black Hawk, Gilpin county, Colorado, at an elevation of nearly 8000 ft. (2438 m.) above the sea.

Like its successor at Argo, it was, primarily, a gold and silver smelter using matte as a collector; and as the entire scheme of the process demanded a high rate of concentration for the immediate production of a matte rich in the precious metals, the ore-mixture could not be rich in copper. Hence, it was not a copper-smelting operation, and those who insist in describing it in that light lose the whole point of the process.

It was strictly a custom smelter and, for metallurgical purposes, it will suffice to divide its ore-supply into three groups, one of which, however, was of a most heterogeneous character commercially, although, metallurgically speaking, it may be regarded as a unit in this description. These three groups are:

(*a*) The so-called "smelting ore" of the well-known Gilpin county gold veins. At the time referred to, smelting charges were very high, and these ores were sorted pretty clean at the mine, and consisted largely of pyrite and chalcopyrite, with small amounts of galena and blende, and a silicious gangue. They might, perhaps, average four ounces in gold and seven ounces in silver per ton, with four per cent copper.[2]

After being roasted in heaps, they were not far from neutral;

[1] My remarks regarding the work at this smelter may be accepted as tolerably accurate, owing to the fact that I lived close to the plant during much of the years 1869 and 1870, as well as during a portion of 1874, and spent much time in studying its practice. I quote no statements from other writers which disagree with my own impressions.

The most accurate and useful description that has ever been published of this plant during its wood-burning era is the paper by the late Professor Egleston, entitled "Boston and Colorado Smelting Works," *Trans. Am. Inst. Mng. Engrs.*, Vol. IV, p. 276.

[2] One ounce per ton = 0.0034286 p. c.

silicious, if anything. This was the class of ore for which the smelter was built, and no idea was entertained at the start of attempting to smelt with wood the refractory ores, high in silica and zinc, which, at the period of which I am now writing, were beginning to embarrass the work.

(*b*) Pyritic concentrates resulting from buddling the tailings from the stamp-mills in which were amalgamated the low-grade silicious ores occurring in profusion with the smelting ore *a*. This material might average one and one-half per cent copper, with one and one-half ounces gold and three ounces silver per ton. This was the chief fluxing material for the entire process, and was bought on such a margin that there was no loss in its treatment.[1]

(*c*) In this class I include the wonderful telluride ores of Boulder county, running up to $10,000 per ton and more, in appreciable quantities, and extremely silicious; surface ores from the surrounding districts high in silver, and averaging 70 per cent silica; beyond all, however, in quantity and importance the rich zinc blende, and galena silver ores of Georgetown, often carrying arsenic and antimony, and always high in silica.

The metallurgical treatment consisted in roasting the Gilpin county gold ores in heaps, while the concentrates (flux) and the sulphide silver ores were roasted thoroughly in hand calciners.

A suitable mixture being made up from the raw and roasted material, the fusion was conducted in wood-burning reverberatories for the production of a matte containing about 30 per cent copper, and carrying some 28 oz. gold and 800 oz. silver per ton, the slag, for the most part, going over the dump. We are not concerned with the disposition of the matte, and may now return to the central figure of this description — the wood-burning smelting furnace.

The preparation of the charge for this fusion shows plainly the importance attached to the fine condition and thorough mixing of the ingredients. With coal as fuel, we may slight this prep-

[1] I have still a record of the sale of concentrated tailings of my own, containing a little more of the valuable metals than the above assay, for which I received six dollars per ton at the smelter. This would seem somewhat unsatisfactory at the present day, but — except as a flux — was of no advantage to the smelter, the high costs and small capacity (about 30 tons daily, including flux) demanding a margin of at least $40 per ton to leave any reasonable profit.

aration to some extent — although usually to the disadvantage of the process — but with the low standard of heat entailed by the use of wood, we must avail ourselves of every reasonable measure to aid the fusion by mechanical means.

The heap-roasted gold ore was crushed through a screen with four meshes to the inch, as were also the oxidized silver ores. The sulphide silver ores, rich in blende, were crushed through a six-mesh screen as a preparation for roasting. All the constituents of the charge were thoroughly mixed by hand before entering the smelting furnace.

The original wood-burning reverberatories at this plant were constructed with an ordinary grate and with no admission of air above the grate. While these answered reasonably well to smelt the fusible Gilpin county gold ores and concentrates for which the plant was designed, they were not found satisfactory when called upon for the higher temperature required for the silicious and zinciferous ores of the neighboring districts. As these refractory ores came gradually to form a larger proportion of the mixture, it became evident that the furnace conditions were unsuitable. The period of smelting a charge was prolonged often for two hours, or more; the slag was thick, and carried globules of matte which would not settle; accretions formed upon the hearth, and required several hours of hard firing to remove them, and all indications showed plainly that the temperature attained within the furnace was too low for the type of slag that commercial reasons required it to produce. Various experiments were tried in the hope of increasing the heat, but without any improvement sufficiently radical to encourage the hope that a reverberatory furnace using the light spruce wood of the Gilpin county hills could smelt successfully the particular types of ore which had come to be the most important source of revenue.[1]

[1] One of the experiments that I witnessed at that time is so peculiarly instructive that it is worth recording. Each furnace had its own separate stack 50 ft. in height (15.25 m.). Feeling that the draft might be insufficient, the management added an additional 16 ft. (4.9 m.) to one of these stacks, but with no perceptible change in its behavior either for better or worse. Few objects are more given to individual idiosyncrasy than the stacks of smelting furnaces, and I should not undertake to base any absolute advice on this single instance. I have, however, the notes of two other cases where I added to the height of individual smelter-stacks connected with wood-burning reverberatories; the stacks were, respectively, 55 and 60 ft.

At this time Richard Pearce became the technical head of the Boston and Colorado smelter, and instituted some changes which, although simple and comparatively inexpensive, improved the furnace conditions to such a degree that the temperature necessary for the comparatively refractory slag was attained without difficulty, and without increase in the amount of wood burned per ton of ore.

The following description, therefore, applies to the Black Hawk furnaces as remodeled by Richard Pearce. I also reproduce herewith, by permission of the American Institute of Mining Engineers, the original plans of these furnaces which accompanied Egleston's descriptions, already cited. Apart from the use of gas-producers, I know of no radical improvements that have been made in wood-burning reverberatory smelters since the period to which I am now referring; and, although these plans represent a design of nearly 40 years ago, I offer them in the belief that no better model can be found to-day.[1]

Referring to the accompanying plans, we find a small reverberatory, having a hearth 15 ft. 7½ in. long by 9 ft. 9 in. wide.[2] The most noticeable feature in the construction of the furnace is the arrangement of the fire-box and air-ports for the more efficient combustion of the fuel, and it is mainly due to this modification that the smelting conditions were improved so greatly.

It will be noticed that the longitudinal dimension of the fire-box lies parallel with the long axis of the furnace. This is not important except in so far as it makes the firing more convenient. The essential feature of the fire-box is its depth, and the tapering of its sides toward the grate, as shown plainly in the cross-section. This is often employed for wood-fired boilers, but I never appreciated its full advantage until I witnessed the results obtained

in height, the former at an elevation of 10,000 ft., the other of about 8000 ft. above sea level. The addition of some 15 ft. to their height had no effect that I could detect. It seems probable therefore that, other proportions being correct, no advantage is gained with a wood-burning reverberatory with separate stack, by increasing its height beyond 55 or 60 feet.

[1] In making this statement I am, of course, referring solely to the general design of furnace and fire-box, and am not suggesting that the metallurgist of to-day should not avail himself of every improvement in details of construction that the coal-burning reverberatory has witnessed.

[2] One foot = 0.305 m. The foot is divided into 12 inches.

by this modification of the ordinary grate. The fire-box is 4 ft. 6 in. high from grate to roof, 5 ft. long, and 5 ft. wide at the top — the latter measurement tapering down to 2 ft. 6 in. at the grate.

There being nearly four feet in depth of burning wood maintained upon the grate, and the latter admitting insufficient air for complete combustion, we have here a sort of gas-producer, delivering a great volume of highly-heated combustible gases. In order to burn these completely, a double row of checker-holes (now so common with coal-burning reverberatories) above the bridge-wall admits air through the roof, and a good admixture is assured by an offset in the arch (see Fig. 76) which causes a whirling of the

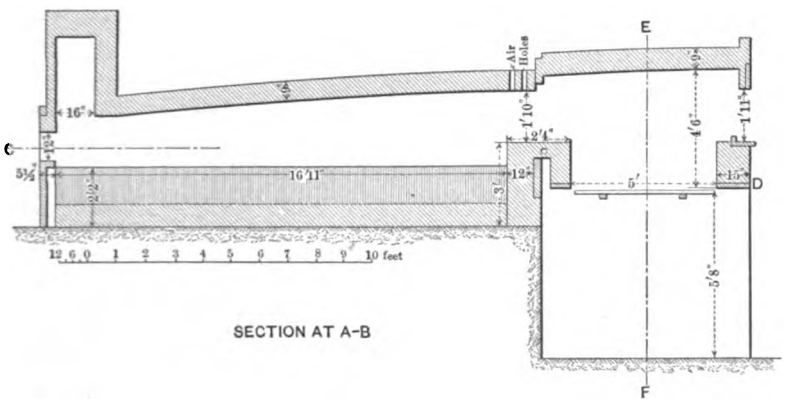


FIG. 76. — Boston and Colorado wood-burning reverberatory — longitudinal section

current at this point. The manipulation of this device for regulating combustion, length of flame, distribution of heat, etc., is now too familiar to furnace-men to demand discussion in this connection.

The general dimensions of the furnace may be useful as showing what long experience proved to be suitable proportions for a reverberatory burning air-dried spruce wood at an elevation of 8000 ft. above sea level.

Metallurgists of the present day will inquire at once whether it is possible to make the furnace of a capacity more in accord with modern ideas, or whether the limitations of the fuel restrict them to something approaching the size here shown. We tread on somewhat doubtful ground when we approach this point, and

I will discuss it more fully after having described the results obtained with Pearce's model, merely stating in advance that it is probable that wood will heat a hearth considerably larger than the one here illustrated without, however, approaching the efficacy of coal as regards size.

The hearth was formed of sand containing a slight admixture of wood ashes and was melted-in in the ordinary manner, although the heat was never all that could be desired for suitable agglomeration. An ordinary charge averaged about four tons of mixed ores,

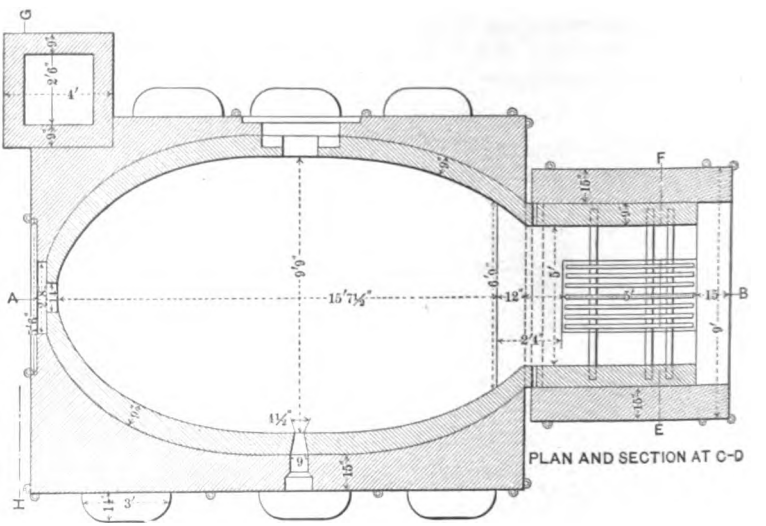


FIG. 77. — Boston and Colorado wood-burning reverberatory — sectional plan

to which was added about 500 lb. of rich slags and sometimes a little limestone or fluorspar. With the improved furnaces, four charges were smelted in 24 hours — or 16 tons per day. Run according to modern conditions, I feel sure that the capacity would have been from 20 to 25 tons per day. This will not seem unreasonable when we reflect that the Black Hawk furnaces were charged with cold ore, shoveled in by hand through the side-doors, which had to be open 15 minutes for the operation. Also that the skimming and tapping were conducted in a manner that caused a much greater loss of heat than our present methods, and that the doors were opened two or three times during each charge to rabble it up from the bottom.

The charge, as already intimated, was a difficult one. It varied extremely owing to the nature of the business, but, after the first few years, was always high in zinc, so far as I am aware. This necessitated a tolerably high percentage of iron to make it as fusible as possible. There are several analyses available from that period, but they are too variable to be of value. The silica content was usually in the neighborhood of 35 per cent, and the zinc ran from 10 per cent up to 15 per cent — and possibly more. These high zinc slags, accompanying so rich a matte, were neces-

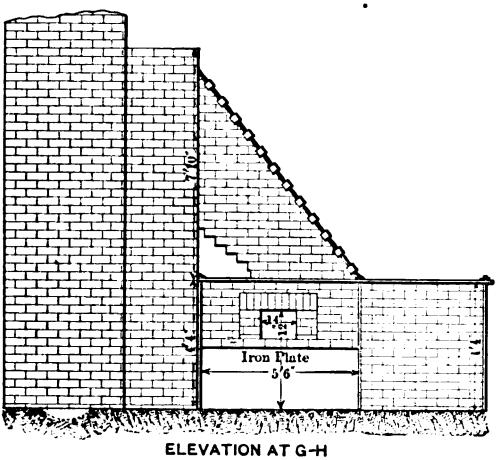


FIG. 78. — Boston and Colorado wood-burning reverberatory — front elevation

sarily high in silver — seven to ten ounces per ton not being unusual.

Returning to the important question of size, we may first gather such information as may be deduced from the behavior of the Black Hawk furnaces. In the first place, the gases left the hearth so hot that one of the great troubles used to be that the fire-brick lining of the stack was burned out with undue rapidity. Under the old system, this occurred no doubt because the gases were not thoroughly burned over the hearth; but even after Pearce had effected unusually perfect combustion by admitting air through the roof, the stack was still too hot until he admitted cold air through a large opening near its base. I know from personal observation that this air did not cause any combustion

in the stack; on the contrary, it simply diluted the heated spent gases and cooled them down to a less harmful temperature. This shows conclusively that, after complete combustion had been effected and the gases had traversed the entire length of the hearth, they were still excessively hot, and would destroy fire-brick unless diluted with cold air.

It seems safe to presume that the Black Hawk reverberatories might have been several feet longer, even without increasing their grate area, while, judging from their appearance while running, I feel sure that their width could also have been augmented in proportion. If, in addition, there had been a corresponding in-

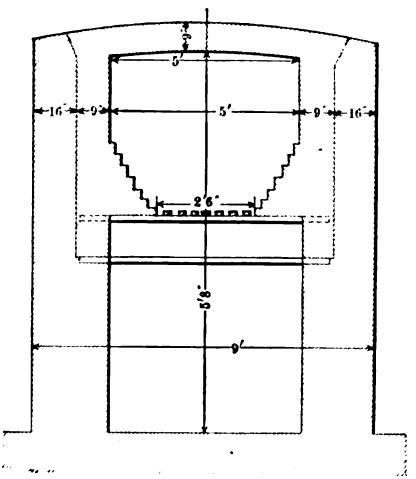


FIG. 79. — Boston and Colorado wood-burning reverberatory — section through fire-box

crease in the area of the grate, it becomes evident that a very fair-sized hearth may be thoroughly heated with the use of wood, the critical dimension being, of course, the length. Later, I shall add some more direct evidence on this point.[1]

[1] I have been corresponding with Mr Pearce on this subject and, since writing the above, I have received a letter from him which has such a direct bearing upon the question at issue that I shall take the liberty of quoting two statements therefrom. Mr. Pearce says: "In 1879 we started a furnace for wood at the Colorado smelter at Butte, Montana, embodying all the principles of the latest developments which were found so successful at Black Hawk. These furnaces were worked with marked success for some years,

I can testify from personal observation that the wood-burning reverberatories at the old Colorado smelter in Butte were doing reasonably rapid work on a very difficult charge consisting very largely of the copper-silver ore of the Gagnon mine, high in zinc blende, and tolerably silicious, even after concentration. These concentrates were roasted, and mixed with custom ores low in copper, but high in silica. The main flux for this acid, zinc-bearing mixture consisted of oxidized manganese ores containing a fair amount of silver, and consisting largely of pyrolusite and psilomelane. The resulting slag was of a very peculiar nature, and had a high formation-temperature as I know from laboratory tests upon it. It is described briefly by Richard Pearce in *Trans. Am. Inst. Engrs.*, Vol. XI, p. 59. "The oxide of manganese plays two parts; it acts as a flux for the silica, and also as an oxidizing agent on zinc blende, which is present in large quantities. The slag produced is of a very peculiar character. According to the results of a late analysis, it contained 30 per cent MnO, 12.5 per cent ZnO, and 48 per cent SiO_2. It gelatinizes both in strong and in weak hydrochloric acid. The slag was very fluid and, when rapidly cooled, it had the appearance of obsidian. Slow cooling gave it a green appearance such as is sometimes seen in lime slags."

The wood-burning reverberatories at Butte, running upon the ordinary copper concentrates, presented no peculiar features, their charge being fusible, and the furnaces experiencing no especial difficulties, excepting that they heated slowly as compared with coal-burners, because of their lower maximum temperature. To the best of my recollection, the wood-burning furnaces at the Parrot smelter, of which I had charge in the early eighties, had hearths 11 ft. 6 in. wide, and about 16 ft. long. They were supposed to smelt a charge of four tons in six hours, with about three

and I am under the impression that, before we changed over to coal, we had lengthened the hearth to close on twenty feet. I would have no hesitation whatever in building a furnace for wood, the hearth to be 20 ft. long by 12 ft. wide, with a corresponding increase in the grate area."

Mr. Pearce also refers to an improvement which had an excellent effect upon the Black Hawk furnaces, and which was added after the accompanying plans had been prepared. This is a device for preheating the auxiliary air that enters through the checker-holes in the roof, consisting of a brick curtain. It is described and illustrated, in connection with another furnace, in the paper of Egleston's from which the other illustrations were taken.

cords of wood, or 0.75 cords wood per ton of ore. This result was attained when everything was running normally, and there were no delays. As a matter of fact, the average daily results for long periods were about 14 tons of charge per 24 hours, with a wood consumption of close upon 0.9 cords per ton.

Before recapitulating the somewhat miscellaneous evidence upon which the wood-burning reverberatory must rest its case, I would call attention to the detailed description by Edwin M. Clark, of a home-made wood-burning reverberatory built in a remote Mexican district, and peculiarly instructive to technical men. As the experience and expedients given in this article would have been invaluable to me in early days when I was in an inaccessible district with plenty of wood and ore which I was required to convert into a matte fit for expensive shipment, I am confident that it will be of equal value to-day for men similarly placed. It also has the advantage of being modern, and thus combining the use of a former fuel with present methods of furnace work.[1]

The material to be treated in the little furnace which is the subject of the following description consisted mainly of pyritous, silver-bearing concentrates carrying a moderate amount of copper, but silicious and high in zinc blende. A thorough preliminary roasting being absolutely essential to success, Mr. Clark built a 14 by 50 ft. hand roasting furnace of stone and adobes, at a cost of about fifteen hundred dollars.

The draft for both the roaster and the smelting furnace was obtained by running a trench up the side of a steep butte, on which a 20-ft. stack was erected. The trench was 300 ft. long, 4 × 6 ft. in the clear, after arching over, and gave a total height to the chimney of 110 feet.

The roasting furnace treated 12 tons of ore per 24 hours, reducing the sulphur from 23.8 per cent to 7 per cent, and using about 2.2 cords of wood. The labor on this furnace, per shift of 12 hours, included seven men, among whom are counted the foreman and the men for handling ore and wood.

The chief interest lies in the little reverberatory smelting furnace, which was more thoroughly home-made than any with

[1] The article to which I refer is entitled, "Smelting Copper in Small Reverberatory Furnaces," by Edwin M. Clark, *Min. & Sci. Press*, April 23, 1910. I make use of Mr. Clark's illustrations, as well as of his description.

which I have ever been familiar. The accompanying illustrations show it in detail:

The hearth was only 7 ft. × 13 ft.; the fire-box, 4 ft. × 4 ft. 6 in., and 2 ft. 9 in. deep below the bridge, auxiliary air being admitted through the arch, as in the Black Hawk furnaces. The stack was 24 in. square, and the flue, 12 in. wide. The red brick for the outer walls, and other unimportant portions, were made upon the spot, as were also the silica brick which formed not only

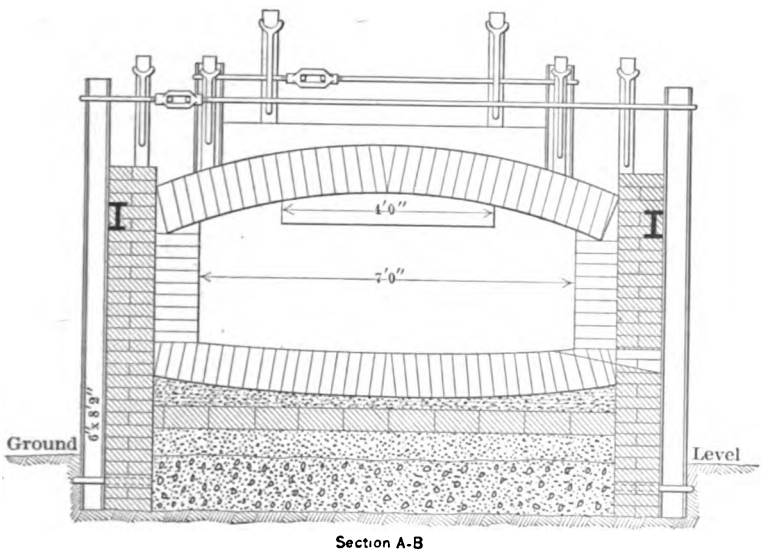


FIG. 80. — Home-made wood-burning reverberatory — cross-section

all the parts subjected to heat, but also the bottom, or hearth. As descriptions of the manufacture of silica brick in this home-made manner are rare, I give the method used by Mr. Clark in some detail, premising that they were somewhat fragile, but stood the heat well.

The materials used were a neighboring clay of a composition far from choice for such a purpose, and some good quartzite, which was crushed dry through a 30-mesh screen. These substances gave the following analyses:

	Clay	Quartzite	Silica Brick
SiO_2	68.77	95.15	90.10
Al_2O_3	16.64	4.15	5.33
Fe_2O_3	3.96		2.67
CaO	2.23	0.60	1.25
Loss by ignition	4.12		
	95.72	99.90	99.35

The manufacture of the silica brick was conducted by mixing two parts of the dry, crushed clay with five parts of pulverized quartzite (by measure) in a mortar-box, adding enough water to make a consistency resembling ordinary lime mortar. When thoroughly mixed it was thrown upon canvas to dry until sufficiently stiff to press into bricks. Many such batches were prepared in advance, so that the pressing might proceed without interruption. The bricks were pressed in a hand press of German make, which molded a brick $2\frac{1}{2} \times 5 \times 10$ in. at the rate of 1200 to 1500 per day. The brick were then carried on a small board to be dried in the open, and were then built into the center of a kiln, the arches and outside walls of which were made of red brick, thus protecting the fragile silica brick.

As is customary with all American reverberatory furnaces, the furnace was built without a vault, starting, in this instance, upon the solid ledge. The outline of the furnace was built up out of red brick, the tie-rods being introduced at the proper level; these being completed, the structure simply resembled an uncovered box of red brick, with an L. The foundation for the brick hearth was prepared with considerable elaboration, consisting of a foot of crushed rock and common clay, slightly moistened and tamped on top of the foundation ledge; upon this was placed six inches of clean sand, followed by a layer of red brick on edge; upon this again was tamped a layer of brasque, consisting of equal volumes of clay, sand, and crushed charcoal, and this was cut and scraped into the exact shape which was desired for the bottom.

With excellent judgment (in my opinion), Mr. Clark decided to smelt upon a naked bottom of silica brick, which were set endwise to form the usual inverted arch. It requires a good furnace mason to lay a tight and regular invert of this nature; but, if it is properly done and the furnace securely ironed, I prefer it to

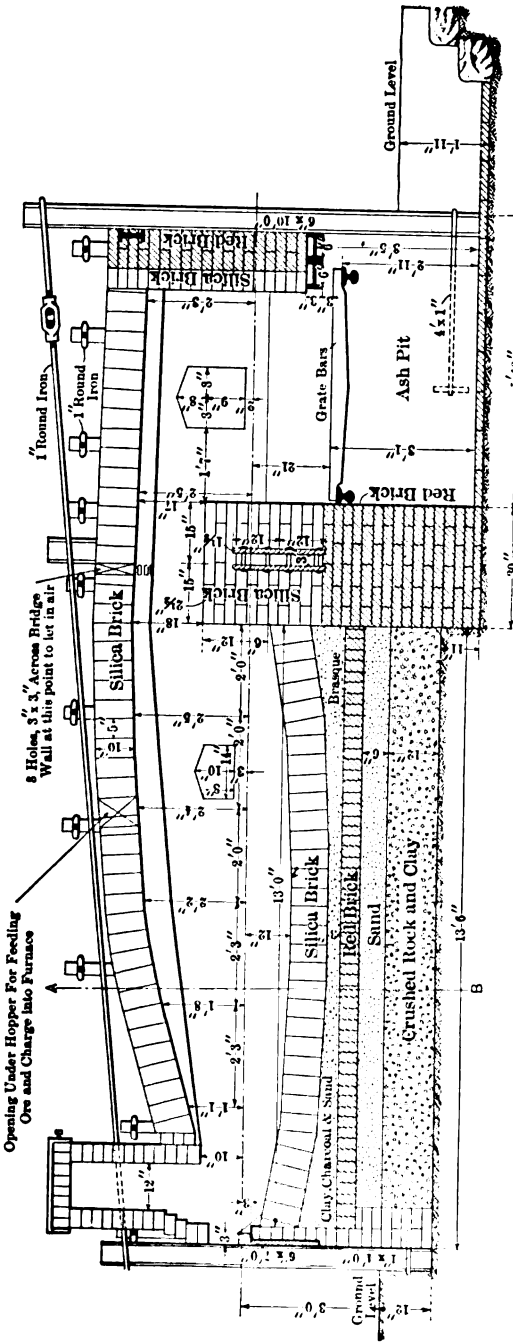


FIG. 81. — Home-made wood-burning reverberatory — longitudinal section

any other kind of bottom, except for very large and long furnaces, where the expansion of the brick may introduce a dangerous factor, and make it safer to retain the customary sand bottom.

After thoroughly drying the furnace, about half a ton of pulverized slag was placed around the edges of the hearth, and the furnace was brought gradually to the highest heat attainable. Leaving the doors still closed, the fire was allowed to smoulder until the hearth was only dull-red, when the first charge of one ton of sulphide ore was introduced, and quickly smelted. Hereupon followed the regular charges of 3000 lb. about every $2\frac{1}{2}$ hours. It required about seven days to fill the hearth with matte, the latter weighing about nine tons.

The roasted ore, which formed the main portion of the charge, had the following composition; the analysis of the slag also being given:

	Roasted Ore	Slag
SiO_2	31.62 per cent	44.10 per cent
FeO	15.34	20.60
Al_2O_3	12.52	16.64
CaO	tr.	5.28
PbO	9.00	3.40
ZnO	7.40	9.10
S	6.10	—
Cu	13.10	0.70
Ag	0.634 (185 oz. p. t.)	0.026 (7.6 oz. p. t.)
	95.714	99.846

The matte had the following composition:

Cu	50.5 per cent
Pb	12.0
Zn	3.2
Fe	4.2
S	25.0
Insoluble	2.1
Ag	2.266 (661 oz. p. t.)
	99.266

About one-half of the charge consisted of hot, roasted ore, which was dropped into the furnace from a hopper, thus saving the time and fuel which is doubly important in wood-firing, but which is so often neglected in these small isolated furnaces.

Mr. Clark makes a few remarks about the behavior of the furnace, a portion of which I quote: "The slag shows the silver

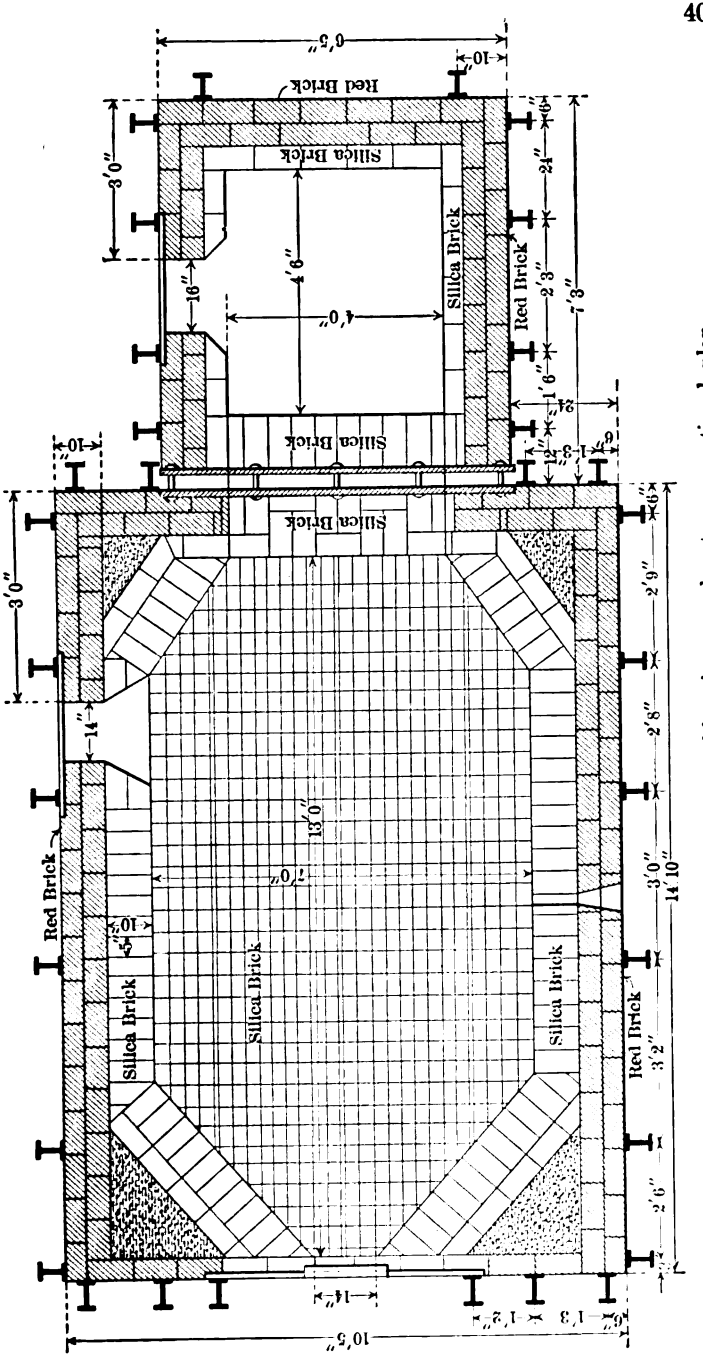


Fig. 82. — Home-made wood-burning reverberatory — sectional plan

to be too high for good work, but this was due to the fact that I did a great deal of smelting with green wood, and it was almost impossible to drive the charge as rapidly as it should have been done at a time when the gases were coming off freely.[1]

"With dry pine wood, I was able to smelt at the rate of three-fourths cord of wood to the ton of charge, and to skim each charge of 1½ tons every 2 hours and 30 minutes. With this small furnace, I smelted 1000 tons of ore in four months, and the furnace was still in perfect condition.

"It was abandoned for a larger furnace, with 10 × 20 ft. hearth, which ran intermittently for three years, floods and Yaqui troubles interfering with steady work.

"In regions remote from transportation, where wood is abundant, if the management will use nothing but *well-seasoned dry wood* (I take the liberty of italicizing this fundamental limitation. — E. D. P.), and has sufficient ore available for long campaigns, the small reverberatory furnace just described in detail can be made to smelt between 10 and 15 tons per 24 hours, with less than ¾ cord pine wood per ton of ore where one-half the charge consists of hot calcines. Any intelligent person who will be careful of his material can make a silica brick which will stand well in such a furnace. I do not, of course, recommend wood where coal can be obtained."

This little Mexican reverberatory presents several features that are worthy of the most careful consideration of men who may be called upon to erect and manage small or medium-sized reverberatory smelting furnaces.

I have already pointed out the cheapness, simplicity, and general advantage of the brick bottom, provided always that it is made of proper material; that the sand or clay or brasque bed for it is absolutely stable, and is shaped to conform exactly to the required invert, so that the bottom-brick can be laid absolutely tight and true, and thus form as strong an arch as would be demanded if it were convex instead of concave; that the furnace be strongly ironed, and never allowed to undercut at the slag line; that it be kept tolerably full of matte, and that the matte be kept

[1] Note Mr. Clark's trials with green wood in connection with my own experience of 35 years ago on the same point, detailed in the earlier portion of this chapter.

protected by a layer of slag (except for the rare intervals when a tap of clean matte is desired), so that the matte shall not "boil"; and that careful watching and patching be maintained.

The deep fire-box and auxiliary air-supply through the arch contributed largely to the success of the smelting. I believe even better results would have been obtained by adding still another foot to the depth of the fire-box and contracting the grate to half its width; also, probably, by slightly preheating the auxiliary air.

The slag analysis given is so unusual that I wonder whether it represented the general average of the work; if so, the wood must have been unusually good, as both the formation-temperature and flowing-temperature of this high silica-alumina-zinc compound, with low iron, come pretty near the limit of what one would ordinarily expect from wood. It is slow work — though perfectly feasible — to make a wood-burning reverberatory deliver a 44 per cent silica slag; when to this is added 16 per cent alumina, the toughness and ropiness of the slag is increased to the critical point; and when, on top of this, we put 9 per cent of zinc oxide (necessarily with low iron), it seems as though we had gone beyond the limits of experience.[1]

Recapitulating briefly our scanty knowledge of former experience in smelting with wood, and combining it with the enormous improvements in the general manipulation of coal-reverberatories which the past few years have witnessed, I will summarize the more important points that the metallurgist should bear in mind when considering the subject.

(*a*) Any kind of wood which will give reasonable results under boilers may be suitable for smelting. Woods, like coals, differ

[1] The space which I have devoted to Mr. Clark's work is the best evidence that I am not doubting his statements; but unusual technical results always provoke discussion, and the comparatively easy fusibility of a slag of this composition would be such a valuable point to establish, both commercially and scientifically, that I hope Mr. Clark will give us further light on the subject. We must always recollect, however, that a slag which would chill at the tuyeres, and thus be impossible, in a blast-furnace may work comparatively well in a reverberatory. So long as it is silicious, even a very tough, viscid slag may permit excellent settling of the matte, and may roll out of the skim-door in a thick, sluggish stream, where it would freeze up a blast-furnace in an hour.

materially in their calorific value. As a rule, the hard woods, such as oak, mountain mahogany, mesquit, maple, etc., are the most valuable, although sometimes the extra labor of splitting them (for seasoning) offsets their better quality. Pinñon, juniper, cedar, and hard pine stand next in order of value. Last of all come the softer pines, spruces, and firs, but even these are quite satisfactory if cut at the proper season, used in proper condition, and burned with proper precautions.

(*b*) No ordinary variety of wood is satisfactory or economical unless properly seasoned. By this, I mean that it should be air-dried until its moisture is reduced (in dry climates) to 20 per cent or less, while it should not be exposed to dry-rot, or to the ravages of the wood beetle.

(*c*) It should be burned in a deep fire-box with contracted grate and auxiliary air through checker-holes in the roof. The firing demands as much care and skill as is exercised in the management of coal. At the best, a wood fire is apt to be too loose and open.

(*d*) It is probable that a hearth 20 ft. long and 12 ft. wide is not too large for ordinarily good wood. I believe that even 14 ft. would not be too great a width, with suitably proportioned fire-box. If the furnace has an individual stack, it seems useless to build higher than 55 ft., as no advantage was noticed in several widely separated plants where the experiment was tried. The area of the stack must, of course, be augmented in proportion to the increased size of the grate.

(*e*) The bottom should be constructed of silica brick, laid by an expert. It should be kept covered with matte, as in the coal-burning furnaces.

(*f*) The complete exclusion of outside air from the hearth during smelting, and the constant closure of the working-doors at all times, should be one of the main features of the work. If this is important with coal, it is indispensable with wood as a fuel.

Under reasonably favorable conditions, we may hope, in such a furnace, to smelt 250 lb. charge in 24 hours per sq. ft. of effective hearth area, with a consumption of three-fourths of a cord of dry wood per ton (1220 kg. per sq. m. with 3 cu. m. per met. ton).

In the seventh edition, and in all later editions, of my old "Modern Copper Smelting" I have published a description, with working drawings, of the generators used for making gas from wood at Atvidaberg, in Sweden, where reverberatories are heated in this

manner satisfactorily and economically with the use of refuse wood. I believed this practice would be valuable in this country, especially in Mexico and Arizona, where much cheap and excellent material exists in the shape of wood which is too gnarled and crooked for an ordinary fire-box, and thus possesses little commercial value.

As I am not aware that these descriptions and drawings have ever been of use to any one, I will not encumber the present volume with their repetition, but will reaffirm that I am more confident than ever of the simplicity and economy of this practice, where suitable conditions prevail.

Those who feel interested in the subject may consult old "Modern Copper Smelting," seventh and all later editions, and will also find some useful practical information in a paper by John Langton, entitled "The Power Plant of the Moctezuma Copper Company, Nacozari, Mexico," *Trans. Am. Inst. Mng. Engrs.*, Vol. XXXIV, p. 748.

The wood-gas was made from air-dried scrub white-oak, weighing about 2200 lb. per cord. Its composition differs materially from producer-gas made from coal, but its calorific effect is nearly identical. I append analyses from Langton's paper which are interesting:

TYPE COMPOSITION OF COAL-GAS AND WOOD-GAS

Components	Coal-Gas volume p. c.	Wood-Gas volume p. c.
CO	20.0	14.0
H	14.0	20.0
CH_4	2.0	2.0
C_nH_m	0.2	0.2
CO_2	8.2	16.0
O	0.1	0.1
N	55.5	47.7
	100.0	100.0
Calorific value per cu. ft. calculated at sea level, and 60 deg. F.	132 B.t.u.	132 B.t.u.

It will be seen that, as compared with coal-gas, the proportions of CO and H in wood-gas are reversed, and the percentage of CO_2 is doubled.

CHAPTER XII

BLAST-FURNACE VS. REVERBERATORY

METALLURGICAL literature contains such an abundance of argument and discussion upon this subject that I hesitate to add to it. My present object is to attempt some simplification of the question, and to narrow the field of controversy to such limits that a very few pages will suffice for such consideration as it can legitimately claim in a text-book of this nature. As in almost all arguments, simplification follows elimination.

If we determine the especial features in which each type of furnace excels, and then allot to each all the cases that are unquestionably suited to its peculiar powers, we shall find that we have substituted order for confusion, and have reduced the subject to a bulk that can be handled with reasonable satisfaction.

It is manifest, then, that the first step in discussing the comparative advantages of blast- and reverberatory-furnaces will be to enunciate clearly the characteristic points in which each type of furnace excels. I shall endeavor to confine myself to statements which are so obvious to experienced metallurgists that they do not need to be supported by argument, pointing out, however, that we cannot, for this particular purpose, include under the term "experienced metallurgist" a man who has spent his life in managing blast-furnaces without ever having become thoroughly familiar with the reverberatory — or *vice versa* — and also recognizing the fact that it is impossible to state any question in such a manner as to satisfy those who approach it in a partisan spirit.[1]

[1] As any writer's judgment upon a subject is likely to be tinctured by his own personal prejudices, I may say that, so far as I can judge, I have no distinct preference for either one of these two great classes of furnace, having — during the active portion of my professional life — employed each, or both, during many years, and having experienced the most lively satisfaction as well as the most profound disgust with each (and both), according to the character of the ore and fuel that I was required to use, and the zeal and ability exercised by my foremen. In the *Mining and Scientific Press* of March 3, 1906, James W. Neill asks twelve questions bearing upon the com-

The first and most fundamentally important advantage pertaining to the reverberatory furnace is its ability to smelt very fine material. The blast-furnace can deal with a considerable proportion of fine material — say down to one-quarter inch (0.006 m.) in diameter — but when it comes to flue-dust, or to concentrates from the fine jigs and the shaking-tables, they cannot, advantageously, be added to the blast-furnace charge without modifying their condition by briquetting or agglomerating — and then they cease to be fines.

These fines, however, in the vast majority of cases, consist mainly of sulphide concentrates, and require roasting before they can be smelted in the reverberatory with the production of a suitable slag and matte; whereas, in the blast-furnace, they not only need no roasting, but form a most valuable fuel, replacing coke (within certain variable, though always circumscribed, limits) to the extent of something like one pound of coke to each pound of sulphur that they contain, or on even more favorable terms provided that the chief sulphide is pyrrhotite instead of pyrite.

Before entering upon the complications which are already beginning to arise, and which will soon make it difficult to set up any exact comparison of the merits of the two systems of smelting, we may at least hold on to what we have already determined, and may enter one valuable point to the credit of each type of furnace.

REVERBERATORY	BLAST-FURNACE
Typically suited to very fine material.	Not suited to very fine material.
Sulphides must be roasted.	Demands no roasting of the sulphides and uses them as fuel.

One of the complications to which I have just referred becomes obvious as soon as we consider the source of all this great amount of finely-divided sulphide material with which the modern copper smelter is required to cope. It results, of course, from the processes of mechanical concentration which have been cheapened and improved to such an extent as to have had a large share in rendering available great quantities of low-grade disseminated ores that could not have been worked with profit a few years ago. In spite, however, of all our modern improvements, the losses in concen-

parative merits of blast- and reverberatory-furnaces. The replies to these questions appear at intervals for a period of several months, and emanate from some of the best qualified men in this country. This discussion is indispensable to metallurgists who desire to study the practical side of the subject.

tration are very great — especially in the case of low-grade ores — running usually from 20 to 35 per cent of the total metal values contained in the original ore.

The question at once arises whether it would not pay better to do the concentrating by fire rather than by water, and thus avoid the greater part of these enormous losses; in other words, whether it would not be more profitable to smelt the entire mass of low-grade ore in the expensive blast-furnace, and thus obtain a high saving of the values, than to submit it to the cheap water-concentration and put up with the accompanying heavy losses of metal. Will it pay better to eliminate the earthy portion of the ore in the shape of a clean, but expensive, slag, or as rich, but cheap, tailings?

This apparently simple question introduces at once the complications to which I have already referred, and which must be thoroughly analyzed and understood before the metallurgist is in position to make any just comparison between blast- and reverberatory-smelting. This complication results from the fact that, if we contemplate a direct smelting of the whole mass of low-grade ore, we must, in the great majority of cases, introduce large quantities of a new barren material into the furnace to serve as a flux for the great amount of gangue-rock, which usually forms 60 to 80 per cent of the low-grade ore. Under ordinary conditions, this newly-added material will be limestone, which is the cheapest, commonest, and most advantageous flux for the blast-furnace smelting of such ores as we are now considering.

While it is most expensive and unfortunate to be obliged to smelt a large proportion of barren material in order to get one's ore melted, it would not seem, at first glance, that it would introduce any serious complications into one's estimate of costs. Assume, for instance, that experience teaches us that, in order to smelt our low-grade, silicious ores advantageously in the blast-furnace, we must not allow the resulting slag to contain more than 45 per cent silica, and that an analysis of the ore shows that, after deducting the iron for the matte, and canceling our bases as far as they will go against the silica in the proportion of 55 per cent bases to 45 per cent silica, we find that there still remains an excess of 25 per cent of silica — or 500 lb. excess silica for each ton of the low-grade ore. As this is to be fluxed with lime, and as we will assume that we have a good limestone at $0.80 per ton which contains 50 per cent of available CaO; and as we must supply

55 lb. of CaO to each 45 lb. of SiO_2 (or $\frac{5}{4}\frac{6}{5} \times 2 = 2.44$ lb. limestone to each pound of SiO_2), the 500 lb. excess SiO_2 which each ton of the ore contains will require $500 \times 2.44 = 1220$ lb. limestone at $0.80 per ton, or 49 cents' worth of limestone flux for each ton of ore smelted.

Even an inexperienced person would see at once that this 49 cents extra cost per ton of original ore is not the only expense entailed by the use of this barren limestone flux. The cost of smelting a ton of ore is determined by dividing the total cost of running the furnace for a given period (say 24 hours) by the number of tons of ore smelted during the same period of time. The furnace does not discriminate between barren limestone and profitable ore; it costs as much per ton to smelt the one as it does the other, but the entire expense of the operation has to be borne by the ore alone, whether it is present in large proportion or small. Thus, using extreme situations for illustrative purposes: a furnace smelts 500 tons of charge per day, and costs $1000 to run. If this 500 tons of charge should be self-fluxing, and require no addition of limestone or other barren material, the cost of smelting would be $\frac{1000}{500} = \$2$ per ton of ore. Next take an unfavorable case where the ore is highly silicious and requires a pound of limestone to each pound of ore; the furnace still smelts its 500 tons of charge per day, but now only one-half of it is ore, so that here we have only 250 tons of material to bear the entire expense of running the furnace, and the cost will be $\frac{1000}{250} = \$4$ per ton of ore, or just double what it was in the preceding case.

It will also be noticed that the ore-capacity of our furnace has been cut in two, and that we shall require twice as large a plant to do the same work. This means a higher capitalization of the enterprise, and the necessity of increased earnings if the owners are to receive a suitable return on their investment.

All this, however, is still tolerably straightforward and apparent, but we have not yet nearly reached the end of the complications that are encountered when we enter upon the consideration of the comparative merits of smelting or concentration as a method for the treatment of low-grade silicious ores.

The next point in the argument that demands consideration is of a totally different nature from those which have thus far been discussed, and pertains to the increased loss of metals in a smelting operation where it is necessary to add any considerable

amount of barren flux. It is familiar to every one that the slag from our furnace operations always contains a certain proportion of the valuable metals of the smelting-charge, which are thus absolutely lost. Other things being equal, this value bears a tolerably constant relation to the degree of richness of the matte; therefore, if the grade of the matte remains constant, the total amount of metal-values lost in the slag will be proportionate to the total weight of the slag produced. An illustration will make this important point more intelligible: a certain self-fluxing ore yields a matte containing 45 per cent copper, with fair amounts of gold and silver, and is smelted in such quantities as to produce 400 tons of slag per 24 hours, containing $0.92 per ton in copper, gold, and silver.

This $0.92 per ton represents 4.5 per cent of the original value of the ore, so that in this smelting process there is a loss of 4.5 per cent. The ore becoming much more silicious, so much limestone is added that the same quantity of ore that formerly yielded 400 tons of slag now produces 600 tons of slag, which all contains values of $0.92 per ton, as before.

In the first case the amount of ore which we were assuming as a basis suffered a total loss of $400 \times 0.92 = \$368$, which was 4.5 per cent of its entire value; consequently, its total value was $\frac{3}{4}\frac{6}{5}\frac{8}{} \times 100 = \$8178$. On the present assumption, however, where the same amount of original ore produces 600 tons of slag, the total loss in values will be $600 \times 0.92 = \$552$, which is $\frac{5}{8}\frac{5}{1}\frac{2}{7}\frac{}{8} \times 100 = 6.75$ per cent.[1]

[1] The entire space that can be allotted to this chapter might easily be occupied in arguments as to the correctness of this estimate of increased loss due to the addition of barren flux. One person might argue that the lessening of the specific gravity of the slag by using so much lime as a base would tend to make it freer of metal values, to which the reply might be made that the increased difficulties of managing the furnace and settler with a high lime slag would fully offset this slight improvement, and so on ad infinitum, with the result that the student would be bewildered by details and would fail to grasp the main principle.

A simple direct statement — not absolutely correct, and to be amended later when it has sunk in — may or may not make some useful impression upon the average student; but an absolutely correct statement, buried in exceptions and complications, will certainly not be of any use to him. Metallurgists who are sufficiently far advanced to detect the flaws in general statements, such as the one just enunciated, will gain little profit from reading an ordinary text-book of this nature.

We have thus far spoken of the more tangible disadvantages which accompany the use of barren flux, and have referred principally to limestone as the ordinary fluxing material in the vast majority of cases. Continuing to speak of the same material, I will mention still another complication attending its use, and one which cannot in advance be determined with such accuracy that we can express it in exact figures and use it in our calculations as to the comparative merits of water-concentration and fire-concentration.

This point is the effect upon the quality of the slag that is produced by too large a proportion of lime. It is not entirely that the augmentation of the proportion of lime in the slag makes the latter too infusible, as this is true only when matters are carried to an extreme. It is rather that, if we have too much lime in the slag, we are crowding out the one substance which, above all others in these cases, renders the slag favorable, the furnace-conditions suitable, and the process profitable: namely, ferrous oxide.

We assumed that, at any rate, our slag should contain 45 per cent SiO_2; there may probably be five per cent of Al_2O_3, and almost certainly as much as five per cent of odds and ends, such as MgO, ZnO, PbO, BaO, K_2O, Na_2O, etc. As our ore was to be extremely silicious, we might easily have had to add enough limestone so that our slag would contain 30 per cent CaO, and if we call a halt, and add up the constituents already determined, as follows:

SiO_2	45	per cent
Al_2O_3	5	" "
Miscellaneous	5	" "
CaO	30	" "
Total	85	" "

we find that there is only room for 15 per cent of ferrous oxide.

This brings up at once the question of suitable slags, which is discussed in a separate chapter, and I will only say here that, while it is possible to run a blast-furnace on a slag containing only 15 per cent of FeO, it is neither cheap nor comfortable, and should only be attempted by experienced metallurgists, and under the most pressing necessity. As a rule 20 per cent is about the minimum limit for the FeO, unless the conditions are exceptional,

and the practice entered upon with a full knowledge of what it entails.[1]

Returning to the main argument, we see that the employment of fire-concentration (instead of water-concentration) of our low-grade, silicious sulphide ores becomes metallurgically inadvisable when so much limestone-flux has to be added as to reduce the FeO-content of the resulting slag below 20 per cent, while it usually becomes financially inadvisable considerably before this limit is reached. In more specific terms, it pays better to submit these ores to the cheap process of water-concentration, putting up with its inevitable heavy losses and making a quantity of finely-pulverized sulphide product that must be roasted, and smelted in the reverberatory, than it does to attempt to lessen these metal losses by smelting the entire mass, in lump form, in the blast-furnace.

When the grade of the ore is high enough so that the loss in water-concentration becomes greater than the expense of smelting so large an amount of gangue-rock in the blast-furnace, it should be sent to the latter department, and the deficiency in slag-forming iron should be made good by the addition of the coarser of the concentrates which result from the water-dressing of the poorer sulphide ores.

This last statement introduces us to a new point demanding consideration, and this time applying especially to the reverberatory furnace.

While the study of the formation-temperature of silicates seems scarcely to justify the comparative exclusion of lime as a flux for silica in the reverberatory furnaces, it is undoubtedly the fact that practical metallurgists object strongly to the use of any considerable quantity of lime in this manner. This objection is not based exclusively upon the natural dislike to smelting barren flux in an apparatus where the ratio of ore to fuel is seldom higher than four to one; it obtains just as strongly in localities where coal is cheap and coke is dear and, so far as I can judge, is founded upon two principal reasons: (*a*) the fact that the addition of any

[1] While the practice at Mansfeld, Lend, Kongsberg, Magistral, Deadwood, Rapid City, and many other similar plants is familiar to the profession as an example of slags much lower in ferrous oxide than the limit which I have just placed, they are all of them distinctly exceptional and difficult, and are to be avoided rather than imitated, whenever possible.

considerable amount of lime to the charge increases materially the corroding action of the slag upon the silica-brick walls that enclose the hearth, and thus compels more frequent fettling with the expensive delays and cooling that accompany this operation; (*b*) the increase in the flowing-temperature (not the formation-temperature) of the slag which results when any considerable proportion of the ferrous oxide is replaced by lime, and which slows down the smelting process considerably. In order to obtain the extraordinary tonnage now so universal in American reverberatory practice, the metallurgist establishes his process in such a manner that there shall be a temperature above the fire-bridge very nearly 400 deg. C. higher than the flowing-temperature of his slag. According to well-known laws this remarkable difference between the temperature of the fresh charge of hot calcines, and that of the portion of the hearth upon which it drops, causes an exceedingly rapid absorption of heat by the cooler of the two substances and, almost before it seems possible that it should occur, the formation- and flowing-temperatures of the fresh ore-charge are reached, though at a degree of heat far below that of the enclosing hearth. The fresh accession of heat arriving constantly from the fire-box raises the temperature above the bridge close to the fusion-point of pure silica — about 1600 deg. C. — and would soon (and occasionally *does*) destroy arch and lining, were it not mitigated by the dropping of a fresh charge of calcines which, although often red-hot, are still cold in comparison with their surroundings. So long as the previous charge softens enough to flatten out in a short time, and float on the matte-bath so that it can be pushed toward the front end of the hearth by a fresh charge, the furnace is cooled with sufficient frequency to prevent undue damage to its arch, bridge, and lining; but when the flowing-temperature of the slag is too high, and the previous charge does not flatten and get out of the way in time to drop a fresh charge before the heat becomes dangerous, serious damage is likely to result. Modern reverberatories are running normally at a temperature close to the danger line, and ten minutes of a heat only a few degrees above that line will do more damage to the furnace than ten times as many hours at a heat only a few degrees below it. A Seger cone will stand erect and strong at a temperature of, perhaps, 1550 deg. C.; but at 1560 deg. it will become a miserable object.

At the present time this condition is illustrated forcibly by the reverberatory smelting of the silicious sulphide concentrates from the low-grade disseminated porphyry deposits. The product of certain of these concentrating mills is practically all finer than one-quarter mm., while much of it is finer than one-tenth mm. It contains 35 to 40 per cent silica, and the addition of sufficient lime to reduce the silica in the slag to a reasonable limit corrodes the furnace too rapidly. Consequently, the slag is run with some 47 per cent of silica, and has so high a flowing-temperature that the process is slow, the cooling effect of a fresh charge recurs too infrequently, and the furnace suffers correspondingly.

Admitting, therefore, that both high lime and high silica are inadmissible for the reverberatory furnace when run on modern lines (and when it is not run on modern lines, its cost per ton becomes prohibitive), we are left with only a single base — ferrous oxide — to serve as our mainstay in operating this process. As most pyritous copper concentrates are pretty nearly self-fluxing — or require the aid of but a small proportion of lime to make them suitable — it would seem that we had here a case of extreme simplicity, and that, if calculations showed that it was not profitable to go to the expense of agglomerating or briquetting our fine concentrates and smelting them in a blast-furnace, it would be a straightforward reverberatory proposition.

But here again occurs one of those complications which render it so difficult to make any positive straightforward comparison between the advantage of the reverberatory and the blast-furnace. Each type reacts upon the other, and the entire comparison is overshadowed and modified by the necessity of accepting concentration and roasting as an essential portion of the question.

In the present instance the complication is caused by the failure to take into account the potential value — if not absolute necessity — of the iron content of the concentrates in the blast-furnace. I am not referring just to its value as a flux for the great excess of silica which we may usually anticipate in the blast-furnace lump ores; the matter is not so simple as that. I am referring to the fact that a certain amount of this iron is absolutely indispensable for the blast-furnace slag, and that we cannot run the process without it.[1] If our charge is too silicious, we cannot

[1] Unless otherwise indicated, I am speaking as though iron oxide and manganese oxides were identical.

go on forever neutralizing it with lime. There comes a point when there will not be room enough left for the 20 per cent of FeO which we decided was an indispensable constituent of our blast-furnace slag, and this iron we must have at any price. As we have just seen, the reverberatory is a glutton for FeO, and will not be put off with any less choice base, except to a very slight extent. Can we then, in a district where advantageous iron is scarce, afford to employ a process where nearly the whole of the SiO_2 in the mixture must be neutralized by FeO when the blast-furnace charge is suffering for the small quota of this substance which is essential to its very existence?

It follows, therefore, that although the reverberatory foreman may point with pride to the enormous capacity of his furnaces, and show by figures that he is smelting his charge at a phenomenally low price per ton of ore, he is accomplishing these results because he is being furnished with the most favorable smelting-mixture in the world; that is to say, one which requires scarcely any flux — so that almost the entire energy of the furnace is devoted to smelting ore instead of barren material — and which also contains for its neutralizing-oxide the choicest of all of our bases — ferrous oxide. It would, indeed, be a pity if a process so favored should *not* be cheap and rapid.

But the manager of the entire enterprise, whose eye is fixed upon the composite result of the whole plant — as indicated by the annual balance-sheet — cares little about high records in the reverberatory department if they are attained at the expense of some other operation, and knows well that there comes a point where a part of this iron that enables the reverberatories to run so easily must be diverted to the blast-furnace if the latter is to run at all. His iron, being diverted to the reverberatories, in the shape of finely-divided sulphide concentrates, he finds that, in order to keep the silica in his blast-furnace slag down to the required limit, he must add so much lime that the proportion of FeO sinks below 20 per cent, and trouble begins.

At the risk of repetition, I must emphasize the fact that the blast-furnace superintendent does not ask for iron in order that he may use it to neutralize his silica; he is not accustomed to any such luxury, and he expects to have to put up with lime so far as the laws of nature (modified by our present practice of running blast-furnaces with a low ore-column) permit. He does not,

therefore, expect any such luxury as an addition of iron ore which shall be basic in character. All that he demands is that he may have a portion of the same self-fluxing pyritous concentrates that the reverberatory is using so lavishly; he does not in any way interfere with the composition of the reverberatory mixture, he only lessens its amount.

It follows then that if the concentrate which, after roasting, goes to the reverberatory, contains even just enough iron to flux its own silica, the blast-furnace manager will set free a considerable proportion of that iron by replacing it with CaO, while this released FeO will be sufficient in amount to add the small proportion lacking to bring the FeO in several tons of the blast-furnace slag up to the required twenty per cent.

Thus it may be necessary to divert to the blast-furnace a considerable amount of finely-divided pyritous concentrate that could be treated to better advantage by roasting and reverberatory-smelting, but whose iron is indispensable for the blast-furnace; so that, even when the reverberatory may seem to be smelting very cheaply, it is actually a highly costly apparatus if it is doing its cheap work at the expense of another department. Moreover, the foregoing remarks show how impossible it is to establish any absolute formula which we may use to calculate the comparative advantage of the two systems of smelting, as each case is so radically modified by its own peculiar conditions.

In situations where it is decided in advance that the enterprise is not large enough to warrant the establishment of both processes, and that it must be limited either to (*a*) blast-furnaces, or (*b*) reverberatories, preceded by concentration and roasting, the following scheme indicates, in a general way, the basis on which the comparison must be established. Assuming that the material to be treated is a medium-grade, silicious sulphide, the first portion of the estimate applies to its treatment direct in the blast-furnace, using limestone as flux; whilst the second part of the estimate contemplates the alternative method: namely, concentration of the ore, roasting the concentrates in mechanical calciners, and smelting the calcines in the reverberatory furnace.

In making such estimates, professionally, it is convenient to start with the specific daily amount of ore that is to be delivered at the smelter — say 2000 tons per 24 hours — and then make a rough estimate of the cost of a blast-furnace plant of sufficient

capacity to smelt this entire 2000 tons per day plus the limestone, or other flux, which must be added to it to produce a proper slag. Then a similar estimate is made of the plant which would be required to treat the same 2000 tons per day by the concentrating roasting-reverberatory system. This is essential — even apart from the necessity of knowing how much capital has to be provided — in order to determine the annual amount that must be allowed for interest on the money invested, as well as for amortization of plant, all of which, naturally, forms a part of the legitimate expense of treating the ore.

Then follows the cost per 24 hours of running the blast-furnace plant, which includes — besides the obvious charges — the cost of fluxes, and also the metals carried away in the slag, charged at the actual value which they possess on the spot in the form of matte. General expenses, interest on cost of plant, and an amortization allowance complete the list. This grand total, divided by 2000 (the number of tons of ore treated per 24 hours) gives the cost, plus the losses, incurred in treating one ton of the original ore by the blast-furnace method, and producing (say) a 40 per cent copper matte therefrom.

The estimate of costs of treating 2000 tons daily of a similar ore by concentrating, roasting, and reverberatory smelting is a little more complicated, because there are three different operations involved, of which only the first one deals with the entire 2000 tons of original ore, whilst the remaining two treat only a diminishing fractional portion of the same. This difficulty is easily met by basing the cost of each operation upon one ton of the original ore. In cases where the grade of the final product (matte) of the reverberatory is to be different from the matte produced by the blast-furnace method, allowances must be made, as the net value of the metals in matte increases slightly with the grade of the matte, and the consequent cheapening of the succeeding operations; in the present instance, this complication is avoided by assuming that the grade of the matte will be the same in both cases — 40 per cent copper.

The charges in this case will include the entire cost of concentration for one day (2000 tons ore) plus the values of the metals remaining in the tailings; furthermore, the cost of roasting the amount of concentrates that result from the 2000 tons of ore; and finally, the cost of smelting the calcines that result from the sul-

phides derived from the 2000 tons of ore. To this must be added the general expenses for the entire plant, as well as amortization and interest on investment. This total sum, divided by 2000, will show the cost of producing a 40 per cent matte from one ton of the original ore. With a properly constructed plant, the losses of metal in flue-dust should be so small that it would usually be an unnecessary refinement to attempt to estimate them separately for each process.

Much space has been occupied in attempting to compare the advantages of blast- and reverberatory-methods in those cases where there is a possible choice between smelting an ore direct, in lump form, in the blast-furnace, or crushing and concentrating it, and then submitting the greatly diminished sulphide product to roasting and reverberatory smelting.

Where concentration is out of the question, or where fuel is dear and the gangue of the ore basic and fusible, the comparison resolves itself into a mere choice between the two processes, and becomes much simpler. In such cases the metallurgist must not allow himself to be too much influenced by the low costs attained in the reverberatory when running with an ample and uniform ore-supply and with plenty of ferrous oxide to form the slag best suited to its requirements. In the majority of instances, the ore-supply will *not* be uniform in character, and there may be seasons of insufficient production, when a portion of the furnace capacity must be shut off. In such cases the blast-furnace is much the superior.

The reverberatory lends itself best to the treatment of roasted sulphides, because it is then enabled to utilize the heat contained in the calcines, and at the same time is provided with ample ferrous oxide. It is also peculiarly suited to the treatment of flue-dust; not that the composition of ordinary flue-dust makes it any better for the reverberatory than for the blast-furnace, but because its physical condition unsuits it for the latter apparatus.

If the reverberatory is furnished with exactly the charge that suits it, and the blast-furnace is regarded as a kind of last resort for all the difficult materials about the mine and smelter, the cost of smelting, per ton of *ore*, is likely to be in favor of the reverberatory. It is well, however, to clear the horizon from time to

time by basing the cost of smelting upon the ton of *charge*, and such an estimate is likely to reverse the figures.

The subject of fines has been discussed at length, and it has been shown that the reverberatory is particularly suited to their treatment, although the power possessed by the blast-furnace to use the sulphides as fuel may often render it wise to sinter, briquette, or agglomerate this pyritous material for this purpose. One or two important points of difference still remain to consider, and of these the matter of fuel stands preëminent, because it forms so large a proportion of the total cost of treatment.

There can be no question that the reverberatory is much more exacting in its fuel-requirements than the blast-furnace. Omitting exceptional cases, there are three classes of fuel that are suitable for it: wood, oil, and coal. Wood is out of the question where large furnaces and rapid smelting are required, owing to its low calorific power, as well as to the impossibility of obtaining it in sufficient quantities. It may, however, be of the greatest value at isolated plants, and at the inception of new enterprises in wooded districts remote from railway lines. Oil, of suitable composition, is an efficient, convenient, and excellent fuel when managed by those who have learned to utilize it properly, but is generally too expensive.

Bituminous, or semi-bituminous, coal is the almost universal fuel of the reverberatory furnace, but is rarely found — especially in the western region of North America — of a quality suited to the exacting requirements of the reverberatory smelting furnace. Mathewson says, "For reverberatories of modern construction, coal with long flame is necessary, and this should develop at least 10,000 B.t.u." Plenty of coals develop 10,000 B.t.u., and apparently generate gases freely enough to yield a long flame, and yet are entirely unfit for reverberatory work. This matter has been considered in another chapter, and I can only say here that the metallurgist who is desirous of starting a reverberatory plant at some new point in the West may look a long time before he finds a coal whose calorific power, flame, ash, and general behavior give him any such results as Mathewson obtains at Anaconda with the excellent "Diamond" coal there in use. I repeat that the reverberatory is extremely exacting as to the quality of its fuel, and that it will yield disappointing results if these requirements are not satisfied.

The blast-furnace is much more elastic in its demands. While it, naturally, cannot produce the same effect with a poor coke that it can with a good one, it can utilize almost any kind of coke and produce a smelting heat with it, requiring, of course, a greater amount of fuel when it is of poor quality. Charcoal, if sufficiently solid to withstand the severe conditions, is not a bad fuel for a small, slow-running furnace, but is usually only worthy of local and temporary consideration. Every wood — if used in solid billets, and prevented from being consumed too high up in the shaft — may replace an important proportion of the coke in an emergency. I do not mean that these inferior fuels are advantageous under any ordinary conditions; I desire merely to emphasize the fact that the operation of the blast-furnace is not disorganized by inferior fuel to anything like the same extent as that of the reverberatory.

Another important matter is the comparative power of the two furnaces to treat a charge high in silica. Quartz is the ordinary gangue-rock of ores and, in the great majority of instances, the metallurgist is overburdened with silica. If he smelts his ore direct, and insists on rapidly-running furnaces, he is swamped with basic flux. If he smelts his ore direct, and economizes on basic flux, his furnaces run too slowly. If he washes out the silica with water, his metal losses are excessive. He selects what he believes to be the least of these evils, but always has too much silica in his charge, and is strongly influenced in favor of the type of furnace that can smelt the most silicious charge to the best advantage.

This advantage is claimed by the partisans of each type of furnace, and I am not at all sure that it is a matter of sufficient importance to occupy space in this book. The truth of the matter is that in attempting to run a very silicious slag in either type of furnace, the smelting becomes so slow that it pays better, in ordinary cases, to reduce the acidity of the slag by the addition of limestone; as this cannot be done, to advantage, in the reverberatory, the blast-furnace scores a point in such instances.

There are, however, exceptional cases where it seems advisable to produce a highly silicious slag, and the question arises which furnace can do it to the best advantage. The reverberatory has a way of avoiding this dilemma by floating off a portion of its silica in the shape of fragments of unmelted quartz ("floaters") which swim upon the liquid slag and are skimmed off together

with it. Where conditions are such that the fusible metal-bearing compounds will liquate out of these floaters, and leave them reasonably clean, this plan may have merits, although it demands great care, hand-skimming, and a general character of work that is not compatible with rapid driving.[1]

Somewhere in the neighborhood of 45 per cent silica seems to be the limit of acidity for liquid slags in reverberatory smelting, save under very exceptional conditions. Indeed, as soon as we overstep 40 per cent of silica, the slag (although its formation-temperature lowers as its acidity increases) becomes slow and the process of smelting is retarded.

Pretty much the same condition of affairs obtains in the blast-furnace. While we find it possible to run slags up to 50 per cent in silica, it is extremely slow, difficult, and precarious work, and is only applicable to those rare cases where there is a scarcity of iron, and it is essential to produce slag high in silica and basic earths. Of the two, the blast-furnace, in the light of modern practice, is more suited to high-silica slags than the reverberatory, for its walls cannot be melted at the high temperature which the slag will require, and also because it is easier to obtain an intense, localized heat from the blow-pipe action of the air upon the coke than it is under the conditions of the reverberatory fire-box.

In older practice the blast-furnace had an enormous advantage over the reverberatory in the matter of fuel consumption — averaging perhaps 8 to 1, as expressed in coke, against 3 to 1 expressed in coal. Recent improvements in the reverberatory practice threatened to bridge this gap to a great extent, if not completely. Not only is the ratio of fuel to ore raised from 3 : 1 up to 4.75 : 1, but such a proportion of the waste heat is utilized for producing power that it has become a grave question, in some localities, whether this furnace may not be able to compete with the blast-furnace on its own ground. The superiority of the latter furnace in this respect, however, has been maintained by reducing the coke and calling in the aid of the sulphide portion of the charge

[1] This used to be practised extensively at Swansea, the slags sometimes containing 60 per cent of silica; in such cases the fused portion might contain 45 per cent, the rest of the silica being in the form of quartz "floaters." This same practice was used by Williams at the Colorado smelter in Butte, and by Charles at the plant of the Montana Ore Purchasing Company. It may be regarded as a useful makeshift.

to take its place. This has raised the ratio of charge to coke in an astonishing manner, 12 or 14 to 1 now not being uncommon, while the advantage is accentuated by the fact that the blast-furnace converts into fuel the compounds which the reverberatory demands shall be decomposed by an elaborate process requiring a costly plant.

In the matter of supplies and repairs the reverberatory is decidedly more extravagant than the blast-furnace per ton of charge actually smelted. In the former apparatus, water-jacketed walls of small extent take the place of the great expense of destructible brickwork. The tools are also exposed to more wear in reverberatory practice.

So far as power is concerned (and recollecting that we are now talking of non-concentrating ore) the reverberatory stands far in advance, requiring usually no blast, while it supplies a large amount of steam-power from its waste heat.

Such comparisons might be continued indefinitely, though with little profit. Each furnace is admirable in its own way, excels in certain duties, and is incompetent for others. Consequently, the two do not admit of comparison.

The subject is further obscured by the frequent necessity of linking the process of mechanical concentration to the reverberatory work.

While it is impossible to evolve any general statement that shall aid a person in choosing between these processes in every situation, it is, as a matter of fact, rarely difficult to determine which type of furnace will best suit any specified set of conditions, provided one is familiar with the general principles discussed in this chapter.

CHAPTER XIII

FINES

THE treatment of pulverized, or finely-divided, material is one of the most serious and important subjects that the copper smelter is called upon to consider, and is becoming more and more urgent with the exploitation of large deposits of disseminated ores, with the improvements in the methods of mechanical concentration, and with the increase in the rapidity of smelting which modern practice demands.

Instead of finding that the problem of treatment becomes simplified by our greater experience and more accurate comparison of results, it is to-day more difficult even than in the past, owing to the discovery that it is practicable to replace much of our blast-furnace fuel by raw sulphides. Formerly it was easy to decide that our sulphide fines — being already in a condition exactly suitable for the reverberatory furnace — should be roasted in a separate process, and then be smelted in the manner indicated; and this decision was strengthened by the great cheapening of the process due to the development of the mechanical roasting furnace.

All this great and increasing class of material is again set adrift by the discovery that its sulphide content will replace a large amount of fuel in the blast-furnace, provided that the physical difficulties of manipulation can be overcome.

In partial pyrite smelting, with coke at $10 per ton, pyrite is worth at least five cents per unit of 20 pounds. Thus, sulphide fines containing 50 per cent pyrite are worth $2.50 per ton for fuel, without allowing anything for their chalcopyrite and other sulphides, of which zinc blende is no small aid in producing heat.

It is painful to any manager to have to build a roasting plant and expend some 40 cents in rendering useless two and one-half dollars' worth of good fuel for the sake of getting his ore into condition to smelt it in a reverberatory furnace, when he has already blast-furnaces on the plant that are suffering for the

iron contained in these very concentrates whose value as a flux he is destroying.[1]

Thus, apart from any argument as to the respective merits of the two great types of smelting furnace, and assuming only that we have a large amount of lump silicious ore that we intend to smelt in the blast-furnace, it is evident that it is of great importance to get our sulphide fines into some condition in which we can utilize them as fuel and flux in this process.

I need hardly say that we cannot make use of them to advantage in the form of a powder. With the wind pressure essential to satisfactory partial pyrite smelting, too much would be blown out of the furnace, and, although this flue-dust may be recovered by suitable appliances, the cost of their construction, upkeep, and service precludes any such practice, especially when one considers the large amount of material that has to be smelted over again and, in part, treated many successive times.

Indeed, the rescuing and re-treating of the flue-dust is but the smaller portion of the evils which result from the use of an excessive proportion of fines in the blast-furnace charge. The rapidity of the smelting diminishes greatly, unprepared material reaches the zone of fusion, the heat rises and becomes diffused instead of remaining concentrated in the smelting area, the hearth chills and the tuyeres crust, while the entire process becomes irregular and unsatisfactory. Incidentally, the grade of the matte lowers and the percentage of silica in the slag increases correspondingly, while any attempt to correct these evils by the addition of extra coke only accentuates them by lowering the standard of the oxidation that is essential to the proper execution of the process.

Every smelter is thoroughly familiar with this train of evils and it is unnecessary to dwell upon them. I may therefore assume without further argument that satisfactory blast-furnace

[1] I do not mean to convey the idea that I should expect his concentrates to serve as a basic flux in the blast-furnace. Few concentrates of this nature contain any more bases than they need to flux their own silica. I refer to the fact that such base as they *do* contain is usually iron, and that they can, therefore, when taking part in the slag-forming reactions in the blast-furnace, spare a considerable amount of ferrous oxide — taking lime in exchange — and thus give invaluable aid in making the slag more ferruginous and fusible than if it were too high in lime, as is too frequently the case when smelting silicious ores in the blast-furnace with limestone as a flux.

smelting cannot be conducted with a charge containing an undue proportion of fines, and may proceed at once to the consideration of methods for converting these fines into a more suitable form for use in the blast-furnace.

In this connection, however, we must also consider fine material which does not contain any important proportion of sulphides, be it oxidized ores, dry ores low in sulphur, or the universal flue-dust.

Omitting, for the present, any consideration of the reverberatory furnace as an apparatus for the *final* treatment of pulverized material, we may consider the more feasible methods of preparing them for the blast-furnace. While rich ores, small scale of operation, abnormally cheap labor, or other unusual circumstances may invite the employment of various curious or cumbersome devices for getting fines into a better condition, the methods which might be considered, under ordinary circumstances, may be classified as follows:

1. Melting
2. Adding to converter charge
3. Briquetting
4. Agglomerating

1. *Melting.* — I have termed this plan "Melting" rather than "Smelting" because I wish to signify that this fusion process is not executed for the purpose of beneficiating the ore, but merely of changing it into a suitable form for the blast-furnace process, while preserving its sulphur and iron content so far as practicable. Thus, the smelting in a reverberatory at Cananea of the flue-dust from the blast-furnaces does not in the least belong in this category, because this fusion is not designed simply to convert the flue-dust into a solid material which will be suitable to return to the blast-furnace. On the contrary, the matte there produced — owing in part to the already oxidized condition of the flue-dust, and in part to the admixture of roasted concentrates — is fit to send at once to the converters.

The employment of a reverberatory furnace simply to melt down sulphide fines quickly into matte, preserving their sulphur and iron so far as practicable, is too expensive a process for the purpose, especially as so low-grade an iron matte is peculiarly fiery and corrosive, and is not a pleasant substance in the reverberatory. I can imagine a situation where it might be worth

while to employ this plan, but I know of no case in practice where it is done habitually. Of course, a quick fusion would remove scarcely more than the feebly-attached sulphur of the pyrite which would escape anyway by sublimation in the blast-furnace, if smelted direct; but as the gangue would probably be silicious, and as there would very likely be no bases present for slagging the silica, the whole operation would be too expensive for its results.

2. *Adding to converter charge.*—This method may work up a small proportion of the highly silicious fines, but is not in the least adapted to serious consideration as an outlet for the great quantity of heavy fine sulphides that usually embarrass the metallurgist. It is now a common practice to try to prolong the life of the acid converter lining by adding a considerable amount of silicious ore during the blow, and a certain proportion of this ore may consist of fines. Again, the material for lining the converter itself is now often composed in part of silicious ores; yet, just at this place, which would seem peculiarly suitable for working up profitably a certain amount of silicious fines, we are often debarred because the fines of a given ore are likely to contain a larger proportion of sulphides than the tough quartzose gangue, and it is silica and not sulphides that one desires in a converter lining.

An exception to this general rule is found in the practice at the basic-lined converters of the Garfield plant in Utah, where the Utah Copper Company's pyritous concentrates, which carry some 26 per cent of quartz, are used as a silicious flux.

3. *Briquetting.*—Under this term I include all the methods where the fine ore-particles are held together by mechanical or chemical means, without having been subjected to any heat sufficient to cause even incipient fusion.

In the vast majority of cases the fine material in a moist condition is mixed with some binding material, and is formed into bricks or blocks or lumps by a machine resembling an ordinary brick press. The further treatment of these briquettes depends upon the nature of the binder, and the amount of expense that it is considered profitable to lavish upon them before they are used at the furnace. This is one of the most important of the methods now in use for consolidating fines, but its economy depends largely upon the nature, cost, or possible profit of the available binding material.

The simplest form of briquetting, naturally, is to mix the fine ore with a suitable amount of water and make it into mud cakes, or let it lie in a heap, from which a sufficient portion is added to each furnace-charge. It is hardly necessary to waste space upon this simple process which, although shockingly imperfect, is usually decidedly better for light fluffy fines and flue-dust than feeding them dry into the furnace. Where the ore is of a clayey nature, reasonably good results may be obtained by this plan, especially if the fines are passed through a pug-mill with the exact proportion of water needed to form a rather dryish plastic mass. This favorable conjunction of circumstances is rare and, when present, might be considered to throw the process into that class where clay is used as a binder.

For most fines — and always for sulphide fines — some binder is necessary to hold together the particles of sulphides or quartz or other substances which have no tendency to cohere. Leaving aside tar, sugar-molasses, iron sulphate, and other rare and expensive substances of which use has occasionally been made, the ordinary binders which the metallurgist has to consider are lime and clay — the one acting for the most part chemically, while the other derives its virtue mainly from its mechanical cohesive power.

The utility of lime springs chiefly from its "setting" power, often heightened by the formation of sulphates, as this is developed only by drying. The manufacture of solid briquettes, where lime is used as the binder, demands special labor, appliances, and time. This, of course, adds materially to the cost of the operation and, with large plants, requires so much space and such an amount of handling and re-handling that it brings a most unwelcome gap into the continuity of the train of operations. It is, therefore, avoided by the large smelters, but finds considerable use amongst the smaller works who cannot afford to add a reverberatory furnace. This is especially the case in dry, mild climates, with cheap labor.

Under favorable conditions, and with reasonably good arrangements for handling and drying, the cost of briquetting with lime may be as low as $1.25 per ton of fines. Raw sulphides — even pyritous concentrates with a due proportion of the grains up to one-fourth inch (0.006 m.) diameter — may be briquetted with lime into good solid blocks, provided sufficient attention is paid

to their drying. It is assumed, of course, that lime can be obtained at a moderate cost.

Five to eight per cent of lime (dry weight) is usually employed, the larger proportion being used in granular fines consisting mainly of raw sulphides. Roasted fines are, of course, much easier to briquette, as the iron is now fluffy and porous and the mixture will also form sulphates which aid greatly in hardening the mass.

One of the gratifying features connected with the use of lime as a binder is the fact that it also brings a (usually) welcome base into the mixture, every pound of lime thus introduced saving the addition to the smelting mixture of something like 1¾ lb. of limestone.

At small mines, having their own little blast-furnace plant, this method of handling concentrates and flue-dust is by no means to be despised, especially where labor is cheap and the climate suitable. For small works it requires no elaborate plant; one man, with an ordinary brick press, and helpers to remove the brick as made, can mold 1000 ten-pound brick per day.

The secret of making good solid briquettes without the use of powerful presses lies in the thorough mixing, kneading, and pugging to which the materials are subjected before molding, It is hardly necessary to point out the better results that will be obtained if every particle of the material to be briquetted is moistened to just the right point and then coated with a suitable proportion of the binding substance. The use of clay as a binding material is much more restricted than that of lime, although its employment lends itself better to rapid continuous work on a large scale. Silicate of alumina is valued greatly by the metallurgist in its proper place, but is not regarded with favor as an addition to the smelting charge; hence, its use as a binder for briquetting fines is mostly limited to those cases in which it forms a portion of the ore itself, or where it contains values in copper, gold, or silver.

The ordinary situation is where the ores themselves carry aluminous minerals — ordinarily, kaolin — which are separated from them during the ordinary process of wet concentration, and of which the very finest portion is eventually recovered in the outside settling ponds, in the shape of slimes (or "slums") containing a considerable amount of the sulphide minerals in so finely-comminuted a condition that they will float away even

with the feeblest current. This situation, as well as the succeeding briquetting operation, is peculiarly characteristic of the concentrator of the Anaconda Company at Anaconda, Montana, and the entire practice forms an excellent illustration of a great briquetting plant employing clay as a binder for raw sulphide concentrates.

There are six slum ponds at this plant, each averaging about 300 by 630 feet, or a total area of something over 26 acres (10.52 hectares). The concentrator treats some 8000 tons daily of the second-class Butte ores, and these ponds receive the final slum waters from the concentrate tanks and Wilfley tables after they have already undergone an elaborate system of settling in the tank-house.

When a pond is full of slums, the stream entering it is diverted to an empty pond, and the full pond is drained as dry as possible. It is then excavated by one of two Lidgerwood traveling cableways, the bucket capacity being five tons. The slum is piled and allowed to dry somewhat, and is again taken by cableway and dropped into a hopper on trucks, whence it runs into the railway cars which convey it to storage bins for use at the briquette plant.

The average composition of the pond slum is shown by the following determination of its more important constituents:

silica	60.0	per cent
alumina	18.0	" "
iron	6.0	" "
sulphur	6.0	" "
copper	2.5	" "
	92.5	" "

silver 1½ oz. per ton.

The excessive fineness of its particles, and the considerable content of kaolin, render these slums extraordinarily sticky and suitable for a binder. Formerly a considerable variety of substances was incorporated in the briquettes, but the advantage of introducing the greatest possible quantity of sulphides into the blast-furnace charge has resulted in simplifying the list, as follows:

pond slums	25	per cent
screenings from first-class ore	25	" "
fine concentrates	50	" "
	100	" "

The ingredients for the briquettes are stored in bins, below which run the conveyor belts which transport them to the pug-mill. Four workmen, with hoes, regulate this operation, two being required for the sticky slimes, and one each for the concentrates and ore-screenings. The conveyors deliver their load into the pug-mills of the briquetting machines, of which there are four, each having a daily capacity of 700 tons. These machines are of the "stiff-mud" type, discharging an endless plastic band, which is cut automatically by wires into 10-pound slabs. About 50 tons monthly of wood-refuse from the carpenter shop is also incorporated in the briquettes.

The finished briquettes are carried by conveyor belts to a long row of hoppers, each of which holds 1250 lb., which is the quantity that it is desired to add to each car of furnace-mixture. There are 36 hoppers to correspond to the 18 cars of the charging-train, so that the alternate hoppers are full, and waiting for the train, while the others are being filled. Assuming that the odd-numbered hoppers are full, the front car of the train is spotted under No. 1 hopper, the remaining 17 cars also occupying situations exactly below the remaining odd-numbered hoppers. A single motion of a lever discharges the contents of the 18 odd-numbered hoppers, which fall into the cars — the latter being already nearly full of the proper mixture, as the briquettes form the last addition to the same.

Owing to the magnitude of the operations, and the care which has been exercised in working out all mechanical details, this is probably the cheapest briquetting of copper ores that is done anywhere in this country.

4. *Agglomerating.*—This is the most important, because the most radical and far-reaching, of the four classes of methods that I have cited as suitable for the treatment of fines to prepare them for the blast furnace. Melting them in the reverberatory into a low-grade matte is too costly. Adding them to the converter charge is too limited in its application. Briquetting is too expensive and too cumbersome for large plants, unless a binding material is available which contains values in copper, gold, or silver.

The term agglomeration, as employed in metallurgy, signifies the sticking together of particles or nodules by partial fusion, and may range from a mere slight softening of the surfaces, suffi-

cient to cause mutual coherence, to a semi-fusion of the mass. As it is often practised upon ores containing substances having widely different melting-points — such as quartz and pyrite — it follows, ordinarily, that certain of the constituents melt during the operation and cement together the infusible particles. It is convenient to extend the definition of the term agglomeration to a somewhat different group of cases where some foreign melted material is used as a cement to bind together the fine particles which it is desired to agglomerate, and which may, or may not, undergo a certain amount of fusion from the great heat of the foreign melted addition. The most familiar illustration of this form of agglomeration is the mixing of flue-dust and converter slag to form a suitable material for the blast-furnace. It follows, therefore, that we have two kinds of agglomeration: (*a*) agglomeration by heat alone; (*b*) agglomeration by the addition of melted material.

(*a*) *Agglomeration by heat alone.* — Perhaps the most familiar illustration of this practice is the involuntary — and much deplored — semi-fusion of a portion of the contents of a roast-heap or stall, which results when too much air is admitted and the process proceeds too vigorously. In roasting coarse ore or matte in this manner, it is often customary to introduce a considerable amount of fines into the heap, provided that local experience has shown that the process will bear it; and when such a semi-fusion occurs, the sulphides liquate away from the less fusible quartz, and agglomerate lumps and fines indiscriminately into a mass often weighing several tons. This is, of course, disastrous to the roasting process, where the object is to remove the greatest possible proportion of sulphur, while in this rapid agglomeration much of the sulphides have melted down without losing any sulphur at all excepting the volatile atom which is driven off by heat alone; but it is a most interesting phenomenon to the blast-furnace smelter of to-day whose aim is to get his sulphide fines into solid form without losing any of the sulphur which will serve as fuel in the blast-furnace. It might seem to him that, if it is so easy to agglomerate his fines by heat without the loss of sulphur when his design is to *remove* the sulphur, he might devise a quick method of doing the same thing when his object was to *retain* the sulphur. That he finds this difficult, not to say impracticable, arises from the fact that he is endeavoring to compare

two radically different operations. In his heap roasting, the reason that his fines melt down so easily as ferrous sulphide is because the heap is composed mainly of coarse ore, between the lumps of which there are plenty of interstices still left for the passage of the air currents essential for the combustion of a sufficient amount of the sulphides to produce the heat required for the fusion of the remainder. Therefore, the fine sulphides present are only incidental to the process, and if present in any considerable amount would block up the interstices so that the heap could not burn at all. The real truth of the matter is that we were roasting lump ore, and such little fines as it could stand got the benefit of the heat that it produced.

This brings us at once to the critical point of the whole subject: when sulphide ores are in sufficiently coarse granules, they may be agglomerated by the heat derived from their own combustion, and without excessive loss of the valuable portion of their sulphur. As the particles become finer, the operation becomes more difficult, irregular, and slow; this slowness of combustion, together with the large surface exposed to the air, favors a gradual roasting rather than a rapid agglomeration, and this roasting of course means a loss of the valuable portion of the sulphur, as well as the oxidation of the metal with which it was combined.

The difficulty increases with the fineness of the particles and becomes soon prohibitive unless we have a considerable proportion of granular material to mix with the smaller fines and loosen up the mass so that air may penetrate it; but as most of this granular material is suitable for use in the blast-furnace charge without any agglomeration, we dislike to submit it to the expense of this extra handling, as well as to the certainty of loss of much of its valuable heat-producing power. It is for these reasons that the operation of sintering or agglomerating cupriferous sulphide fines has not yet obtained the foothold that one might expect from the apparent simplicity of the operation and the admirable quality of its product.

It is scarcely necessary to say that there can be no question of attempting to sinter a layer of these fine, or even granular, sulphides without a forced blast of air of sufficient strength to make its way between the heavy, close-lying particles. Without this aid we could not attain the rapid combustion of a portion

of the sulphides sufficient to produce the heat required to yield a partial fusion of the remainder. A slower current of air would simply mean roasting without fusion, whereas our main desire is to produce fusion without roasting — or rather without any more roasting than is absolutely essential to generate the necessary temperature. The blast may be used either in an upward direction by being blown through the layer of sulphides, or in the opposite direction by being sucked, by a fan, down through them. The former practice is the more common and, from the fact that the process was at first carried out in large pots, like those of the Huntington-Heberlein sinter-roasting for lead ores, was termed pot-roasting. The later term "blast-roasting" seems more suitable, as this refers to an essential feature of the operation rather than to the mere shape of the vessel in which it is executed, and which has now ceased to be characteristic.

In 1895, or possibly earlier, M. Louis Pelatan, of Paris, submitted to my examination plans of a stall for the roasting of sulphide fines.[1] It was a long, narrow, covered stall, with a tight iron front and a close-meshed grate. The granular ore, with finer on top, was placed upon the grate in 10-ton charges, ignited from below and, with the aid of light under-wind, was roasted and sintered in 10 hours. It was used at the Laurium mines in Greece on lead-zinc ores, and at various copper sulphide mines in Chili, and correspondence elicited that it gave satisfactory results when the workmen controlled the heat properly. I felt a good deal interested in it at the time, as the disposition of fines has always been one of my chief trials; but I found on further examination that, while it gave excellent results if a sufficient proportion of the charge consisted of granular material, it became slow, difficult to manage, and generally unsatisfactory just as soon as the proportion of strictly fine material was increased beyond a certain moderate limit.

We have more suitable apparatus for the purpose at present, but I call attention to this roast-stall for two reasons: first, because we ought not to forget Pelatan's work in connection with the development of the blast-roasting of copper fines; second, because the same difficulty that hinders the development of the process to-day was the principal reason why Pelatan's stall did

[1] See my old "Modern Copper Smelting," seventh and later editions, p. 173.

not obtain wider recognition years ago: namely, the impossibility of obtaining good results when the charge consisted of too large a proportion of very fine material, combined with the fact that it is exactly this extremely fine sulphide material that we desire to treat.

The process of blast-roasting sulphides lived and died before the successful application of the same idea to the treatment of galena ores, and it is not by following too closely the measures found suitable for the latter material that the copper smelter will succeed in adapting it to his own peculiar wants. I call attention once more to the fact that what we desire in copper smelting is an agglomeration or sintering of our fine pyritous sulphides, without any more oxidation than is absolutely essential to produce the high temperature which is needful to stick the ore-particles together with sufficient firmness to prevent the crusts crumbling into fines before they are landed safely in the smelting zone of the blast-furnace. This means a short intense period of oxidation, and any plan which does not effect this purpose is unsuited to present conditions.

Copper smelters in various parts of the world have been quick to appreciate the advantages of such a method, and many of them have tried it — often on a considerable scale, and with ample expenditure for the best appliances that present knowledge can afford. Their results, however, have been unsatisfactory when judged by the standard just laid down. This need not, however, discourage us, as most of the operators have directed their efforts to *roasting* and sintering the fines rather than to *sintering* them without roasting. Consequently, their results cannot be accepted as conclusive by the partial pyrite smelter whose aim is to conserve his metallic sulphides and not to destroy them, and the details of their work are not sufficiently valuable to be told at length in this place. However, it is important to see what devices have been employed, and what results obtained, even in this unsatisfactory type of blast-roasting, and I will give one or two brief illustrations.

Hofman[1] describes the operations of the Detroit Copper Mining Co. of Morenci, Arizona, where a mixture of 80 per cent sulphide concentrates and 20 per cent of flue-dust was blast-

[1] "Recent Progress in Blast-Roasting," by H. O. Hofman, *Bulletin 42 of the Am. Inst. of Mng. Engineers*, June, 1910.

roasted in spherical pots with results that were satisfactory from a metallurgical standpoint, but were not considered to be advantageous commercially.

The pot swings upon trunnions in a steel frame, being tilted by a differential chain-block to dump the sintered cake. The accompanying illustration shows the details of this pot and indicates the general construction of such vessels.

Each pot treated 10 tons at a charge, the mixture consisting of eight tons of concentrates and two tons of flue-dust. A screen analysis of the concentrates gave the following results:

On ½ in. opening	3.3	per	cent
Through ½ inch and on 10-mesh	9.6	"	"
Through 10-mesh and on 40-mesh	30.9	"	"
Through 40-mesh and on 80-mesh	25.5	"	"
Through 80-mesh and on 120-mesh	14.0	"	"
Through 120-mesh and on 200-mesh	7.8	"	"
Through 200-mesh	8.9	"	"
	100.0	"	"

The average analysis of the concentrate was:

SiO_2	14.3
Fe	24.9
Al_2O_3	5.0
CaO	0.5
MgO	0.5
S	32.2
Cu	18.8
	96.2

The moisture in the concentrate as used averaged 8.5 per cent. The flue-dust had the following composition:

SiO_2	23.9
Fe	25.9
Al_2O_3	5.5
CaO	1.9
MgO	0.5
S	16.2
Cu	17.1
	91.0

The plant having no roasting furnaces to supply a primer of hot ore, a small wood fire was kindled on the ash-protected grate

of the pot, and about 75 lb. of sawdust was added in such a manner as to be about six inches deep at the center and to taper toward the periphery. Then two tons of warm (80 deg. C.) flue-dust was added, and covered by a three-inch layer of concentrate. A blast of two ounces for 30 minutes served to ignite the concentrate thoroughly, when the blast was raised to 18 oz. and concentrate fed as rapidly as the heat rose through it. This charge required about 20 hours for completion, a mixture of half flue-dust and half concentrate being finished in 14 hours.

The best results yielded 93 per cent of coarse material, the analysis of which yielded:

SiO_2	17.6
Fe	34.5
Al_2O_3	5.8
CaO	0.8
MgO	0.6
S	9.5
Cu	21.4
	90.2

At the Wallaroo Smelting Works, South Australia, experiments were conducted on the application of this method to the roasting of the pyritous concentrates from the ores of the Wallaroo and Moonta mines. As the results were favorable, the process was installed, and about 2000 tons (2240 lb.) of concentrates per month is now roasted in five pots.[1]

The following points were established by the preliminary laboratory experiments, and it is interesting to note that they correspond with the results of experience on a large scale:

1. The addition of lime is not necessary to bring about sintering.

2. With suitable ore-mixtures, it is only necessary to start the reaction with a small shovelful of hot coals, cover these with a layer of moist fine ore, turn on a light blast, and then fill in the charge. Desulphurization proceeds steadily till the mass is completely sintered. (It will be noted that, at these works, the object is a combined roasting and sintering instead of the sin-

[1] In the following description I have made use of a paper by S. Radcliff, entitled "The Pot Roasting of Copper Ores," *The Australian Mining Standard*, Dec. 2, 1908.

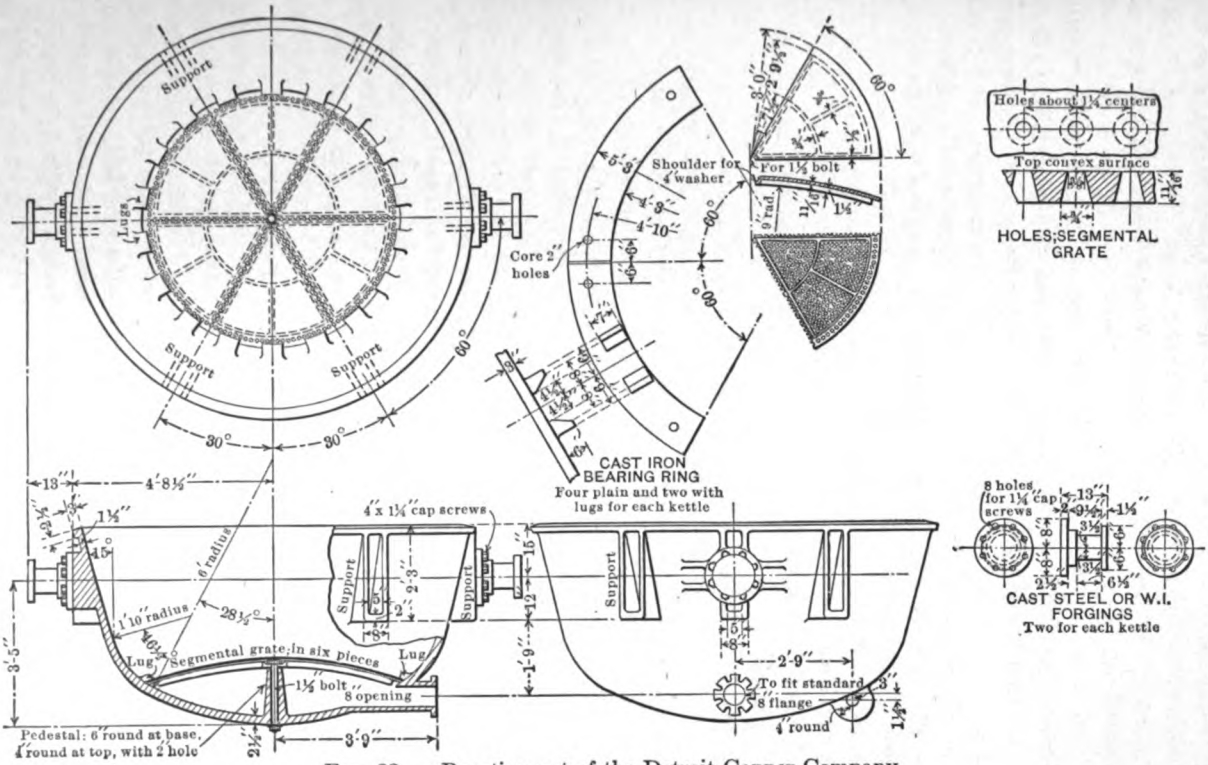


FIG. 83. — Roasting-pot of the Detroit Copper Company

tering with as little roasting as possible which most American conditions now demand. E. D. P.)

3. No preliminary heating or partial roasting of the ore is needed.

4. The presence of several per cent of water is essential. With dry ore, the reaction soon stops, and it is noticeable that a considerable amount of ferric oxide is formed.

The passage of the zone of reaction upward through the charge may be recognized by thrusting an iron rod into the mass after it has been burning for some time, as the presence of a well-defined layer of viscous sulphur is at once evident. This results from the sublimation of the feebly-attached portion of the sulphur belonging to the pyrite and chalcopyrite.

Radcliff points out also that the low heat-conductivity of the charge is an important factor, as it confines the reactions to a horizontal zone only a few inches in depth that rises steadily to the surface as desulphurization proceeds. The water, which is so essential a portion of the charge in blast-roasting, plays a part that has not yet been thoroughly determined. It is essential to prevent dusting; but it seems probable that it also exercises an important influence upon the reactions themselves.

Perhaps the most important point of all to bear in mind in practical work is that in the zone of reaction the chemical changes must progress with such rapidity that a temperature sufficient for the formation of ferrous silicate is maintained; otherwise, there will be a considerable formation of ferric oxide, with local depression of temperature and a resulting patch of unsintered material. Once formed, the ferric oxide cannot be reduced and slagged.

At Wallaroo the size of the charge ranges from $1\frac{1}{2}$ in. diameter, down to fine slimes. It is found that good results cannot be obtained if the charge contains above 40 per cent of slimes, and the usual mixture consists of one-third each of some coarse $1\frac{1}{2}$ in. material, fine concentrates, and slimes from the Wilfley tables.

The pots hold eight tons each, and demand no especial description. They are made of cast-iron and, though inclined to crack, are easily repaired by riveting wrought-iron patches on the outside.

The grate of the empty pot is covered with a layer of the sintered product broken to three-inch size. A wood fire is started

with a light blast, and spread over the bottom by means of a layer of sawdust. Then, about six inches of charge is run in, the blast is increased, and as soon as the reactions are well started, the pot is filled at once, instead of charging in successive layers as is the commoner practice.

In this process, even more than in most other metallurgical operations, the success of the result depends upon the care and skill of the attendants. Vents for the free sulphur are provided by pricking the charge with an iron rod, and these have to be closed at the first sign of the formation of a blow-hole. If a bad blow-hole forms, the blast pressure becomes weakened, and a hollow cylinder of sintered material extends rapidly to the surface. The only remedy when such a condition is established is to shut off the blast, bar out the core, and fill the cavity with wet raw ore.

Only a small amount of unsintered fines remains on top of the completed charge, and this is added to the next pot. Radcliff gives the costs for a typical month's run, based on the ton of 2240 lb., as:

working and blowing charge	$0.50 per ton
breaking and handling	0.16 " "
power	0.37 " "
management and maintenance	0.14 " "
total	1.17 " "

Matte, of any grade, crushed to one-half inch size, and mixed with sufficient silicious material to slag the iron, is worked in this manner at Wallaroo without difficulty. The main points are to have the mixing very complete, and the charge well wetted down.

At the same plant, regule (very high matte; practically, cuprous sulphide) is crushed to half-inch size, well wetted down, and treated in the same manner without the addition of any foreign substances. It is converted into metallic copper without fusion of the charge, as a whole, and many hundreds of tons have been thus treated at a cost of about $2 per long ton. One of the main difficulties in the treatment of this material is connected with the overwhelming evolution of SO_2 gas.

While the process of blast-roasting in pots lacks the continuity so essential to the economical course of large operations, and while

its practice thus far has linked the waste of the fuel-value of the sulphides with the agglomeration of the fines, it is, in spite of these drawbacks, a most valuable resource for many smaller smelters who — equipped with blast-furnaces, and unwilling to erect a reverberatory with its attendant roasting plant — are embarrassed with an excess of pyritous fines from the concentrator.

For such situations, blast-roasting in pots offers a method of treatment which furnishes the iron flux so sadly needed for the silicious lump ore of the blast-furnace department, and also conserves at least a decidedly helpful amount of the fuel-value of the sulphides — an amount which can undoubtedly be greatly increased by care and experience.

Moreover, it accomplishes all this with less expenditure for plant and buildings than any other method with which I am familiar. The advantages of the plan are augmented decidedly by cheap labor and a reasonably dry climate; consequently, it seems peculiarly suited to conditions in Mexico and parts of South America.

One of the most satisfactory applications of this method to such conditions, and one that is especially instructive on account of the large proportion of fine material treated, was installed by Mr. Henry F. Collins at a small copper mine in Spain, and I feel justified in devoting some attention to this practice, believing it to be a typical illustration of work which will be of direct practical value to metallurgists who are connected with similar enterprises.[1]

The materials for agglomeration at this Spanish mine consist of (*a*) concentrates and middlings from both wet and dry concentration processes, carrying 12 to 18 per cent copper as mixed; (*b*) fine screenings from rich ore, carrying 8 to 12 per cent copper.

The following analyses show the composition of the materials treated, and of the product that they yielded after sintering.

[1] The following description is, virtually, a brief abstract of a paper published in *The Mining Magazine* of September, 1909, entitled "Sintering of Copper Ores in Spain," by Henry F. Collins, of England. The ton used is the long ton of 2240 pounds. It must be especially noted that Mr. Collins's aim was to remove sulphur as well as to sinter his fine ore. I think it is fair to assume that a simple agglomeration of the fines, with as little oxidation as possible, would augment considerably the capacity of the plant and decrease the cost per ton correspondingly.

Each of the determinations represents averages of one or more months, but as the roasted products do not represent exactly the same periods as the raw materials, they do not correspond precisely:

	Dry Dressing		Wet Dressing		Crude Fines	Sintered Product	
	Concentrate	Middling	Concentrate	Middling		Ordinary	Exceptional
Insoluble	25.0	50.7	23.0	41.6	60.2	45.5	56.2
Fe	22.3	5.6	24.2	16.7	6.8	18.1	9.8
MnO	0.4	0.3	0.4	0.3	—	—	0.2
CaO	2.4	6.5	2.8	3.4	4.2	7.8	6.5
MgO	0.4	1.0	—	—	—	—	1.2
Al_2O_3	4.2	5.2	—	—	4.7	—	7.1
Cu	20.0	9.3	18.6	10.2	12.5	14.1	7.7
S as sulphide	12.8	6.0	27.8	15.2	7.0	—	2.0
SO_3	3.3	2.8	nil	nil	nil	—	1.4
Total sulphur	14.2	7.1	27.8	15.2	7.0	5.3	2.6

The wrought-iron conical pots are shown in Fig. 84, while Fig. 85 gives details of the perforated grate and cast-steel wind-box. Excepting the pedestals and bearing, hand-wheels, and perforated grate, which are of cast-iron, and the wind-box which is of cast steel, all parts of the pots and trucks are made of mild steel.

Blast is derived through a six-inch branch from an underground main of 16-inch diameter, capable of serving 10 pots. The fume is carried off by means of a hood above each pot, depending from a three-foot sheet-iron flue, which delivers into a fire-brick flue connected with a square stack of red brick. The internal lining of flue and stack is laid in a mortar composed of sand and boiled coal-tar, and the exposed surfaces of the brick are soaked in coal-tar as a protection from the acid fumes. The iron hoods, downtake, and flue were thoroughly soaked in boiled coal-tar before and after erection, and after 16 months' continuous service show little evidence of corrosion.

The capacity of each pot is about 3500 lb. of the silicious concentrate mixture. The pot being run under the hood, combustion is started by a few embers from the boiler fire, on which are thrown a couple of shovels of coal. A light blast is turned

on, and the burning fuel is covered with 100 lb. of the moistened ore-mixture, more being added if the blast breaks through at any point. After 10 minutes' blowing, the blast-joint is broken and the pot run back to the loading-platform, where the full charge is rapidly shoveled in. The pot is then run back under the hood, and blast is turned on at a pressure of seven to eight ounces. Care is necessary at first to prevent the heat from becoming too high at the bottom of the cone just above the wind-box, and it is generally necessary to play a jet of water for a few moments on some portion of the iron where there is a localized red-heat. A number of holes is punched around the top of the charge to ensure an even distribution of the blast, each being choked as it shows symptoms of becoming a blow-hole. Toward the end of the operation the blast pressure rises to 12 or 13 ounces per square inch.

The only preparation of the ore before charging it into the pot is to moisten it thoroughly and uniformly, this being done at the same time that the various classes of fines are being mixed. The proportion of these various classes depends mainly upon the quantity of each which happens to be available, taking care only that there is a considerable admixture of jig-concentrate and middling of over one-fourth inch in size. While not essential, it is found convenient to add to this mixture about five per cent of fines from the oxidized iron ore purchased as flux. When the charge-mixture contains a preponderating amount of fines below one-eighth inch in diameter, so that a decided resistance is opposed to the passage of the blast, it is advantageous to add a small proportion of coarsely crushed slag — not exceeding three per cent in practice — which shortens the roasting period and facilitates agglomeration.

With a considerable proportion of material of one-half to three-quarter inch size, the sintering operation can easily be completed in 6 to 7 hours, but in the absence of coarse material it takes longer; thus, when all the charge passes a one-half inch and nine-tenths of it a one-fourth inch screen, from 9 to 10 hours' time is required for a thorough sintering. Under normal conditions the unsintered portion of the charge does not exceed 5 to 10 per cent of its weight, but when the sulphur is low and the silica high, some 25 per cent is unsintered, and has to be returned to the next charge.

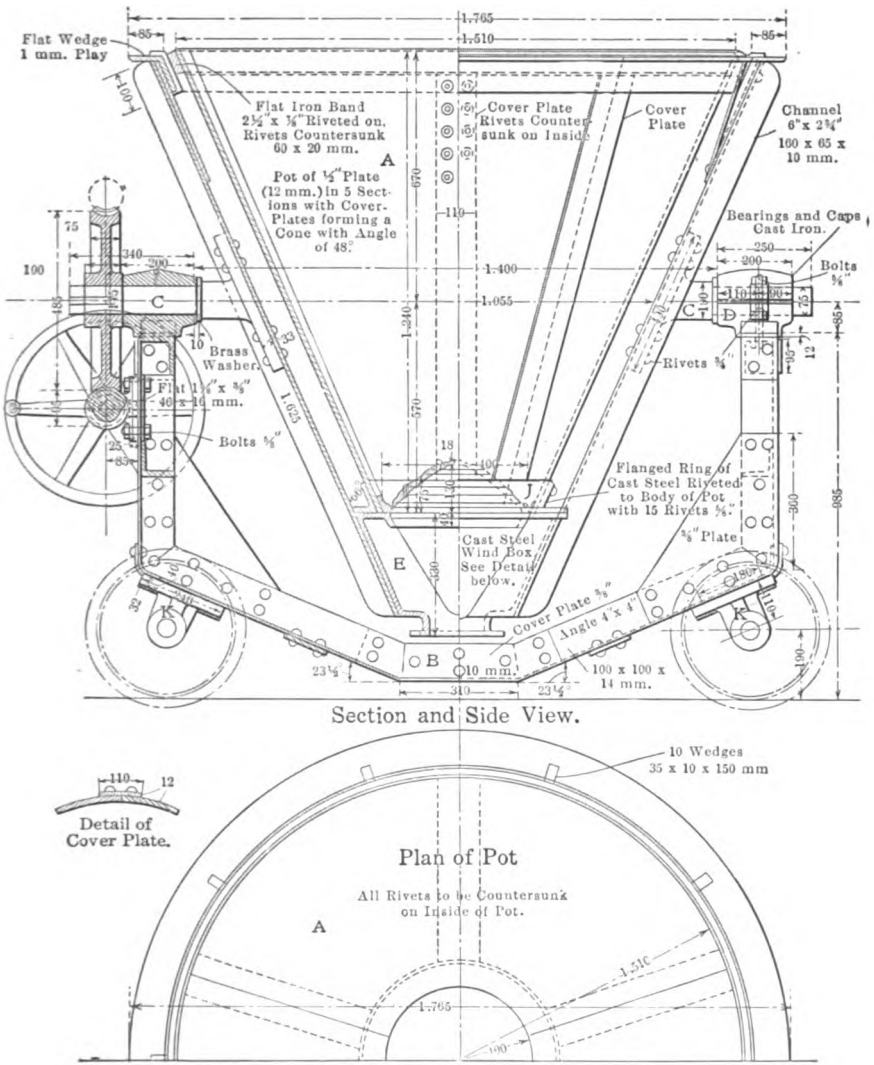


FIG. 84. — Collins sintering-pot

The cone of sintered material dumped from the pot can be broken readily by sledges and picks into lumps the size of a man's head, and is sent to the blast-furnace, for which it forms an ideal material, its physical condition being intermediate between coke and boiler-clinker.

There are only seven pots in commission, and these are worked on eight-hour shifts by four men per shift, the greater portion of the labor being expended upon the breaking and loading of the sintered masses. Each shift is expected to charge, blow, and dump each of the seven pots, separating any unroasted material, breaking the sinter, and tramming the product to the smelter bins. This should mean the charging, on each shift, of 10 tons of material, or a daily duty of $5\frac{1}{7}$ tons per pot; but as some charges take longer to finish, while others contain a certain amount of unroasted material that has to be returned, the average output in actual work is about four tons (8960 lb.) per pot per day.

The cost for labor is about $0.54 per long ton. The blast and wear of tools adds to this about $0.07 per ton, making a total of $0.61 per ton, without allowing for repairs, renewals, and general expenses. The total cost of the installation ready for work was inside $5000, and Mr. Collins expresses the belief that, under Spanish conditions, a large plant using crane-handled 10-ton pots could easily bring the cost below $0.25 per ton. He adds that his experiments have shown that, when mixed thoroughly with a considerably greater amount of raw concentrate fairly high in sulphur — especially when containing a good proportion of coarse concentrate or middlings — flue-dust can be agglomerated advantageously by this method.

I believe the above illustration to be infinitely more valuable to the average metallurgist than general descriptions of large plants, which are not in the least suited to the peculiar conditions under which the process of blast-roasting is likely to assert its greatest value and which, indeed, have for the most part been given up in this country. I will, however, mention briefly one or two instances of this application of blast-roasting on a large scale, in order to show the general arrangement of the plant which was believed to be most suitable for the purpose.

The Garfield smelter, in Utah, installed an extensive plant for the blast-roasting of fine concentrates, comprising 25 pots of

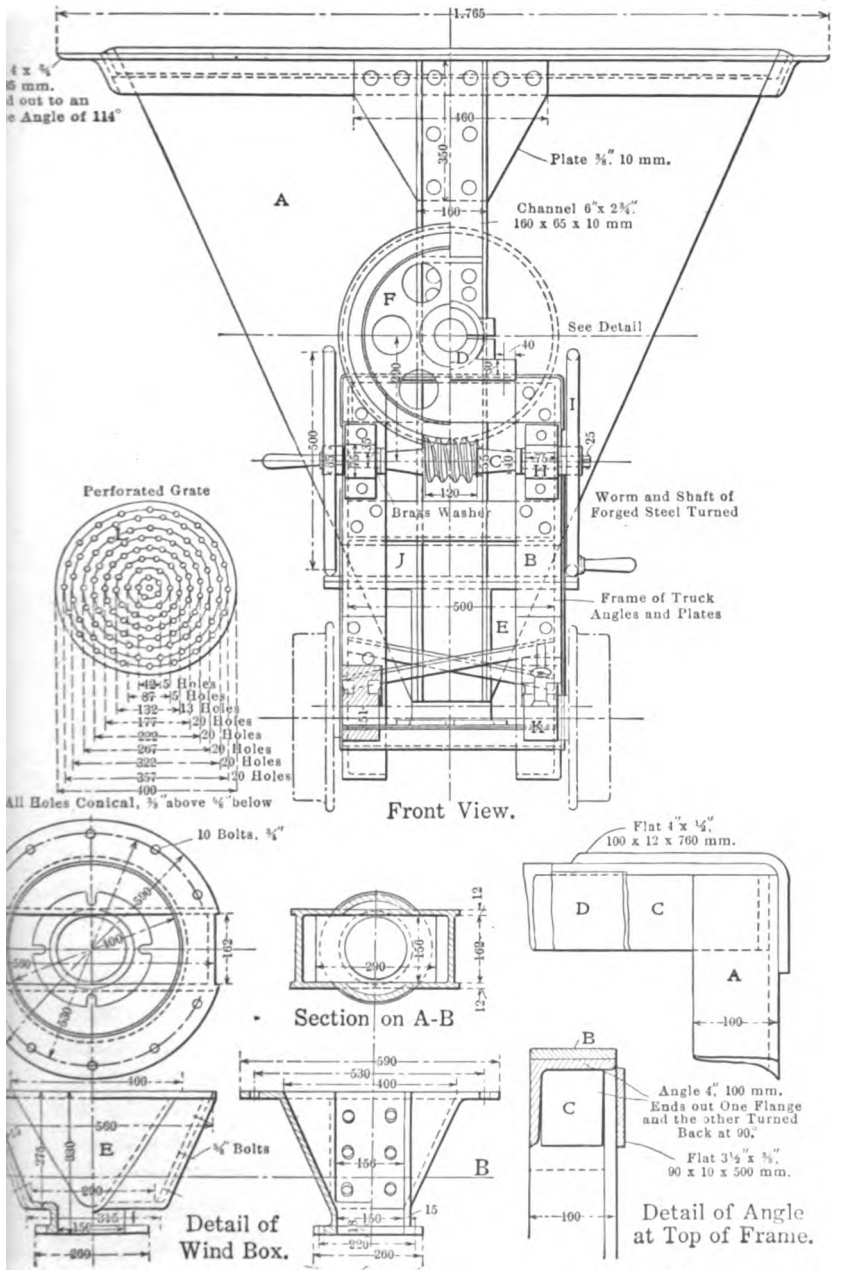


Fig. 85. — Collins sintering-pot. Details of grate and wind-box

the Huntington-Heberlein type. The pots are 8 ft. 8 in. in diameter and 4 ft. 2 in. high, and consist of a top and bottom portion bolted together. The pots stand in a single line, and each pot, when in action, is covered by a sheet-steel hood which conveys its gases to the main flue. The pots hold about eight tons of ore, and are served by a gantry-crane.

In starting an empty pot, the false bottom is covered with an inch layer of ashes, and 2000 to 3000 lb. of partly roasted ore, hot from the McDougall calciners, is added. The blast is then started, and six or seven tons of cold, moistened, partly-roasted ore completes the charge. As will be noticed, the entire charge has already undergone a partial roasting in the McDougall furnaces, the sulphur having been reduced to about 18 per cent. Under normal conditions, the charge is completed in eight hours, the hood is removed, and the pot conveyed by the crane to the end of the yard, where it is dumped upon a stone platform. The large lumps are broken to six-inch size by a 24 by 36 in. crusher.

This practice has now been given up at the Garfield works, for the reason that the results obtained were not commensurate with the care and expense required to turn out a satisfactory, well-sintered product. The experience here coincides precisely with that of every other similar plant of which I have knowledge: namely, that when there is a sufficient amount of coarse or granular material in the charge, the operation is rapid, regular, and yields a well-sintered product with but a small amount of fine stuff to add to the next pot, but that, as the fines increase, the operation becomes slow, irregular, requires constant attention to avoid blow-holes, and yields an excessive proportion of unsintered product and an undue quantity of flue-dust. The concentrates treated here resulted largely from disseminated ores, and were too fine for the process. The partial roasting in calciners was intended to prevent the premature agglomeration of the sulphides, and accomplished its purpose to a considerable extent, but the entire charge was in too finely divided a condition for the process. If this early agglomeration with a minimum of oxidation could be accomplished, it would be just what is required by the metallurgist who desires to sinter his fines without impairing their fuel-value more than is absolutely necessary. I am not aware whether trials of this nature were instituted at this plant.

The investigations of Hofman, as well as the impressions of ordinary observers in practice, show that the sintering of a charge in the blast-pot is a progressive action, starting at the point of ignition and moving slowly upward until it runs out at the top. We learn, moreover, that this action is remarkably localized and circumscribed, so that any given particle of ore is exposed only a minute or two to the maximum heat of sintering (1000 deg. to 1200 deg. C.). It follows, therefore, that the ordinary deep roasting-pot is more of a storage-bin than of a roasting or sintering furnace, because at all times the greater portion of its space is occupied either by particles which have already undergone the ordeal and are dead, or by particles waiting for the active zone to creep up far enough to engulf them. As these layers of inactive particles — respectively above and below the zone of action — check the passage of the blast and cause serious mechanical annoyance, it would seem more reasonable to dispense with them entirely, and to concentrate effort upon the active portion alone. If this could be accomplished continuously, and without stoppages for charging and discharging, stopping blow-holes, and similar annoying interruptions, it would bring the process of blast-roasting into harmony with the modern practice in other departments of the smelter.

The Dwight and Lloyd sintering machine is designed to effect these purposes, and, being in successful operation for roasting and sintering lead ores, leads us to hope that it may be found equally suitable for copper fines. That it is capable of producing an excellent sinter from ordinary pyritous copper fines has been demonstrated on a large scale at various works.

The Dwight and Lloyd sintering machine is based upon the principle of "down-draft": that is to say, of sucking the blast down through a layer of ore, by which all movement of the particles is hindered, and the ignition of the entire thickness of the layer is effected. The continuity of the process is maintained by constructing the grate upon which the ore rests in the form of a large cylinder, or of an endless traveling belt, or a revolving table, and devices are provided for distributing the pulverized ore upon this grate continuously, igniting it, and discharging the sintered product.

The three types of sintering-machines above indicated are known as the straight-line machine, the drum-machine, and the

horizontal-table machine. All embody the same[1] three functions: (*a*) a layer of ore is spread mechanically about four inches deep on a traveling herring-bone grate; (*b*) the ore-stream thus formed passes first under an igniter, placed at right angles to the grate, to become ignited at the surface, and then over a stationary suction-box which, by down-draft, causes the roasting started at the surface to progress downward, and to be completed when the ore reaches the farther end of the suction-box; (*c*) the sin-

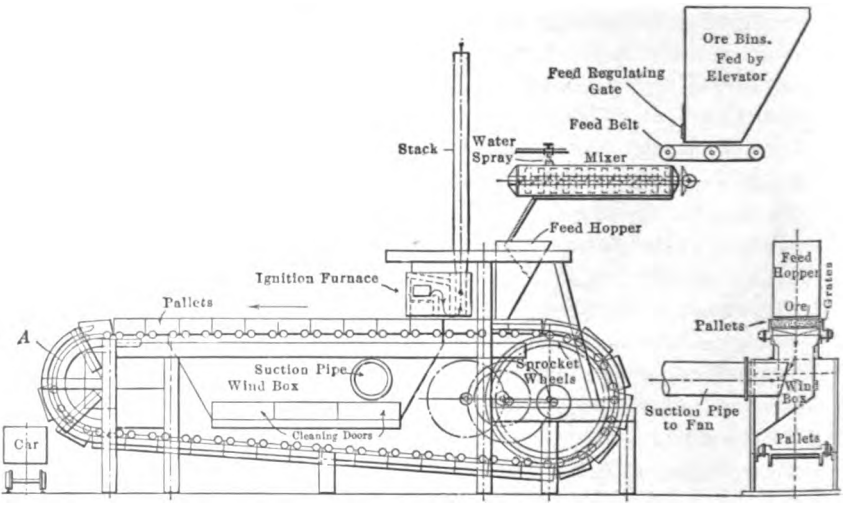


FIG. 86. — Dwight-Lloyd straight-line blast-roasting machine

tered ore is discharged automatically in a size suitable for blast-furnace treatment.

(*a*) The straight-line machine, shown in Fig. 86, consists of a frame of structural steel supporting a feeding-hopper, an igniting furnace, a suction-box, and a pair of endless track circuits to accommodate a train of small truck-like elements called pallets which, in combination, form practically an endless conveyor, with the continuity broken at one place in the circuit. Each pallet is provided with four wheels, which engage with the tracks or guides at all parts of the circuit except when the pallet is passing

[1] Much of the succeeding description is adapted from a paper entitled "Recent Practice in Blast-Roasting," by H. O Hofman; *Bulletin 42 of the Am. Inst. of Mng. Engrs.*, June, 1910.

over the suction-box, when the pallet slides on its planed bottom over the planed top of the suction-box, thus making an air-tight joint. A pair of cast-steel sprocket wheels, turning inside of concentric guide-rails, lifts the train of pallets from the lower to the upper track by engaging their teeth with the roller-wheels, and launches each pallet in a horizontal path under the feed-hopper and igniting-furnace, and over the suction-box. After a pallet passes over the suction-box and terminal dead-plate, its wheels engage the ends of the circular discharge-guides. These are adjusted with the view of raising the pallet about 0.5 inch vertically, and thus automatically prying up the cake of sinter and freeing it from the grate-slots. On reaching the curve of the guides, the pallets one by one drop into the guides, each strikes the pallet which has preceded it and, at the same time, discharges its load of sinter-cake, and shakes free the slots of the grates, the force of the blow being regulated by the gap left in the train of pallets at this point.

The igniter frequently used with this machine is a small coal-burning furnace built of tiles, having a grate area of 10 by 30 inches, and burning 500 lb. coal per 24 hours. The flame, after passing the fire-bridge, is deflected downward upon the ore by a brick curtain that can be raised or lowered, and then is drawn upward by the natural draft of a small stack or bleeder.

The suction-box on top is 12 ft. 6 in. long and 30 in. wide, and gives for the grates an effective hearth area of 31.25 sq. ft.; this is the true measure of the capacity of the machine. The pallets are 30 in. wide by 18 in. long and weigh, with grates, 550 pounds.

The complete cycle of operations is as follows: a pallet, being pushed out tangentially from the top of the sprocket wheels, passes under the feed-hopper, where it takes its load in the form of a continuous even layer of charge say four inches thick, passes next under the ignition-furnace where the top surface is kindled, and at the same time comes within the influence of the downward-moving currents of air induced by the suction-draft; these carry the sintering action progressively downward until it reaches the grates. The roast-sintering operation is complete, the cake is discharged by dropping into the discharge-guides, and the pallet crowds its way back to the sprocket wheels, is slowly raised to the upper tracks, and begins a new cycle.

A straight-line machine of the size described, with effective

hearth area of 31.25 sq. ft., weighs, without accessories, approximately 16 tons.

As regards the power required to drive this apparatus, a machine treating 35 to 40 tons per day of pyritous copper ores will require, for mixing of the ore and propulsion of the pallets, four to six horse-power; for the fan, 12 to 20 horse-power; or a total of from 15 to 26 horse-power. One man per shift can attend the machine.

The main obstacle to the use of a continuous machine of this type is the lost time due to the stopping-up of the grate-perforations, and to the considerable repairs attendant upon the installation of any new automatic apparatus that has to be exposed to a high temperature.

(*b*) The drum sintering machine is well known to metallurgists, having been described and illustrated fully during the past few years, and being in operation at various plants.[1]

The accompanying illustration gives a general view of the machine.

It consists of a horizontal cylinder 11 feet 4 inches in diameter and 3 ft. face, made up of a pair of circular rims of iron carrying cast-iron herring-bone grates with an effective width of 30 inches; the drum-shaped structure rests upon two pairs of friction-rollers, one of which is connected with the power. Inside the drum is a stationary suction-box which occupies the top quadrant of the circle. The ore, fed mechanically upon the rising grate, is ignited by gasoline jets, travels over the suction-box in about 20 minutes, and is discharged automatically from the grate-surface by the points of an unturned grizzly.

Dwight states that the revolving drum consumes from 0.75 to 1.5 horse-power; the fan, from 10 to 20 horse-power; the mixing, elevating, and conveying apparatus, from 4 to 5 horse-power, making a total of from 15 to 25 horse-power.

While adapted especially to lead ores, this machine is now installed at the works of the Baltimore Copper Smelting and Rolling Co., Maryland, for treating, at intervals, mixtures of sulphide copper-ore concentrates and flue-dust at the rate of 25 tons per 24 hours.

(*c*) The horizontal-table machine resembles a horizontal

[1] *Eng. and Mng. Journal*, March 28, 1908. *Mineral Industry*, Vol. XVI, p. 380 (1907).

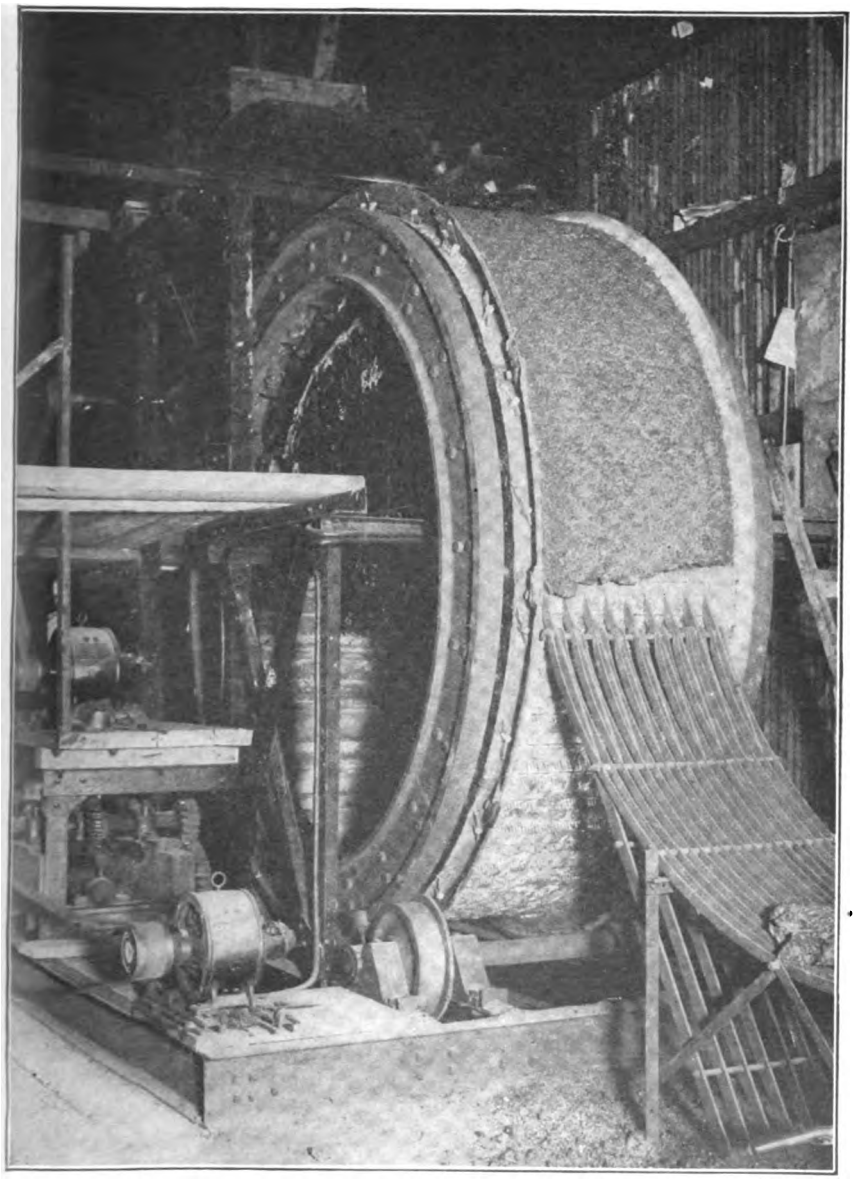


FIG. 87. — Dwight Lloyd drum sintering machine

rotating picking-table in which the ring-shaped table is replaced by herring-bone grates. The outer diameter is 15 ft., the inner 8 ft., giving a total grate area of 126 sq. ft., of which about 50 per cent is effective. The table makes one revolution in about 45 minutes. There is a stationary feed-hopper and igniter, and the sinter is removed by means of a scraper and a deflecting apron. The capacity is large, but the scraper in removing the sinter forces an undue amount of fines through the grate-slots. One of these machines is running on fine copper concentrates (of which 45 per cent will pass a 200-mesh screen) at the Garfield smelter in Utah.

With this machine, as in all other forms of blast-roasting, it is absolutely essential that the constituents be intimately mixed and uniformly moistened, the amount of water varying from five to ten per cent, according to the character of the ore.

The essential chemical composition of the charge will, naturally, vary according to the nature of the process, and the object which it is desired to attain.

In roasting lead ores, where one of the chief needs is to eliminate sulphur, it is necessary to be much more particular about the composition of the mixture than in treating copper fines, where agglomeration is the essential feature and where, indeed, we often desire to have as little desulphurization as possible.

Where roasting is desired, Hofman suggests the following requirements: "The charge-components ought to be so apportioned as to furnish a slag that forms at a low temperature, and requires little superheating to become fluid. The range of SiO_2 lies between 10 and 35 per cent; it is better to have FeO in excess of CaO than vice versa; Pb can be high or low: charges with 60 per cent Pb are being used successfully." The lowest permissible limit of S may be placed at 10 per cent, and there is no limit at the other end in cases where sintering, and not desulphurization, is desired.

A few words in recapitulation may be useful to indicate the position now occupied by the blast-roasting of fines in the repertory of the copper smelter.

At the very outset he is confronted with the fact that when he talks about the blast-roasting of copper fines he is speaking of two essentially different operations: one of them being the

case of sintering the ore with the removal of as *much* of its sulphur as is possible; the other being the sintering of the ore with as *little* removal of its sulphur as possible. The latter I believe to be much the more important case for our modern blast-furnace practice where sulphides replace coke to a very definite extent. It is unfortunate that we have records of but little experience in this more important division, but laboratory experiments — on a tolerably extended scale — seem to indicate that the only serious difficulty that will be experienced in sintering fine sulphides with a minimum of oxidation is the invariable difficulty of endeavoring to make an air-current penetrate excessively fine material.[1]

Assuming that the metallurgist is desirous of conserving the fuel-value of his sulphide fines during the sintering operation, he has at his disposition two distinct methods of procedure: he may use the up-draft, deep-layer method of roasting in pots, or the down-draft, shallow-layer method of the mechanical sintering machines. Until the latter apparatus has proved its fitness for this peculiar variety of sintering without roasting (so far as practicable), it will, no doubt, be wiser to adhere to the cumbersome and non-continuous pot method.

The measure of success — other things being reasonably suitable — will depend mainly upon what proportion of his fine sulphides are sufficiently coarse to remain upon a one-fourth inch mesh screen. If he has some 20 per cent of this size, and possibly 30 per cent more that will remain upon a one-eighth inch mesh, he may perhaps be able to stand the other 50 per cent in the shape of table concentrates, flue-dust, and other very fine material. Or, by adding a moderate proportion of barren or valuable coarse material, such as crushed slag or other granular sulphide ore, he may render a charge permeable that would, otherwise, be hopeless.

The above precaution, together with thorough mixing and uniform moistening of the charge, are the most essential points so far as the material is concerned. The process itself must be accorded all the care and attention that would be bestowed upon the blowing-in of a new blast-furnace, and it is not until the

[1] For a more extended discussion of this important matter, see an essay by the author, entitled "Some Future Problems in the Metallurgy of Copper," *The Mineral Industry*, Vol. XVII, p. 281.

workmen become thoroughly skilled that this personal interest in the process can be relaxed.

(*b*) *Agglomeration by the addition of melted material.* — The utilization of the large amount of rich slag produced by the bessemerizing of copper matte is such an evident means of agglomerating fines that it has long appealed to copper metallurgists, and has probably been tried, after some fashion or another, at every converter plant in this country, not excepting the first one.[1]

There is no question of the ability of the slag to incorporate a considerable proportion of fines, and to yield a mixture that is entirely suitable for the blast-furnace. We have the adhesive material in abundance, with an almost unlimited supply of the powder, in the shape of flue-dust and fine concentrates; the only difficulty is to evolve a convenient, rapid, and economical means of mixing them together. A slight review of some of the more promising methods that have been tried for this purpose may be of use to those who are interested in this matter.

At the Copper Queen smelter, Arizona, elaborate experiments were made upon the best method of agglomerating flue-dust and converter slag. The most satisfactory results were obtained by the use of a slightly inclined revolving cylinder, into one end of which the slag and dust were introduced. The fines and slag became agglomerated into hard balls, varying in diameter from one-half inch up to six inches, and the results were sufficiently favorable to make the process worthy of serious consideration.

At the Sasco smelter, southern Arizona, a considerable quantity of flue-dust and fine concentrates is agglomerated with

[1] I can speak with some positiveness on this point, having been superintendent of the Parrot smelter at the time Manhes' pupils were erecting the first converters at the Parrot works. Noticing the ease with which spilled converter slag picked up the granules of matte with which the floor was covered, my blast-furnace foreman remarked upon the possibility of applying this idea systematically to the treatment of our roasted concentrates which troubled us so much in the blast-furnace. It was tried in a small way, but the development of the bessemerizing operation overshadowed less important work, and I do not think it was prosecuted further. In about 1872 a German foreman in my employ at the Mount Lincoln smelter, Colorado, systematically mixed the extremely rich fines from the surface ores of the Moose Mine (near Fairplay, Colorado) with the ordinary blast-furnace slag, being shocked by the amount and assay of the flue-dust that he collected on the roof of the furnace building. He told me that he had learned to do so at some small smelter in Germany where it was a common practice.

converter slag as a regular portion of the process at that plant. Old matte-ladles are used for mixing-troughs, and the flue-dust and concentrates are drawn into them from hoppers, while the converter slag is being poured in. The two substances, if properly fed, mix together very well, forming a black porous mass containing little fines. I am informed that the slag will yield a satisfactory product with something more than an equal weight of fines. If the process continues to prove as satisfactory as present reports indicate, it would certainly seem an advantageous method of handling this fine material, especially as it does not diminish the effective fuel-value of the sulphides.

At Cananea, Mexico, a method of mixing converter slag and fines has been devised [1] by Ricketts, which yields a satisfactory product for the blast-furnace, and is said to equal in economy the very cheap Cananea reverberatory smelting of flue-dust. "A shelf was excavated in the side of the slag-dump to give a foundation for a slag-car about 14 ft. below the upper track, and the same height above a receiving-bin at the bottom. On the track-level above are provided a hopper for receiving flue-dust, and a hopper with a spout for conveying molten slag to the slag-car below. A spout also extends from the dust-hopper to a point over the center of the slag-car. While slag is being poured and is forming a vortex in the molten mass in the bowl of the car, the flue-dust is discharged into the center of this vortex, where it is drawn down into the slag and becomes thoroughly mixed with it. When the slag-bowl is full, it is dumped over the steep side of the slag-pile, down which it rolls as a viscous mass. As it tumbles downward, it balls up and forms so-called bombs, not unlike the balls of fritted cinder in a cement-kiln, except that they are of larger diameter, ranging from one-half inch to 12 inches. The red-hot bombs are sprayed with water from a hose, being quenched as coke is quenched when withdrawn from the oven. The bombs are of good tenacity, bear handling well, and are porous and permeable to the blast. The converter slag will take up about 20 per cent of either flue-dust or raw concentrates. The FeS_2 present is largely reduced to FeS, thus adding to the speed of smelting."

[1] See "Cananea Furnace Practice," by Courtenay DeKalb; *Mining and Scientific Press*, July 2, 1910.

Bormette incorporates lead or copper fines with a predetermined proportion of slag by rapid stirring of the mixture in a pot. A porous spongy product is obtained, and the operation may be so managed as to produce a considerable degree of desulphurization, or the reverse.

At some plants, mechanical means are employed to effect this mixing, but so far as I am aware, the process has been found cumbersome and expensive compared with some of the preceding methods.

CHAPTER XIV

THE PRODUCTION OF METALLIC COPPER FROM MATTE

WITH the elimination of the original ores of the mine, and the substitution therefor of a new artificial substance, the study of the metallurgy of copper undergoes a complete transformation.

Hitherto we have occupied ourselves mainly with the oxidized portions of the ore, and have been chiefly concerned in regulating their proportions and conditions so that they should melt together into a silicate which should be of such a nature as to permit the separation of the heavier valuable constituents, and flow out of the furnace. These valuable constituents claimed but little of our attention beyond making suitable provision for their removal from time to time. They took care of themselves, because they melted at a low temperature and separated without difficulty, provided the oxidized portion had been managed properly.

The condition is now precisely the reverse. The question of slag is now entirely subordinate. Our matte, indeed, consists of substances which will not form slag, nor yield gases by the application of heat alone. All that heat can do to it has been done already; so that we are now dealing with a substance which is neutral and inactive when acted upon solely by heat.

Before studying the methods employed for freeing the copper from the sulphide grasp of the matte, it will be useful to look a little more closely into the nature and composition of this peculiar substance, limiting the examination strictly to such points as have a direct bearing upon practical metallurgy.[1]

While we have no difficulty in determining with great precision the proportion of each element present in a given sample of matte, we are not yet fully instructed as to the manner in which these elements are grouped. Consequently, some of the statements which I am obliged to make regarding the composition of

[1] See Chapter X in "Principles of Copper Smelting" for a more detailed study of this subject.

matte may be open to criticism. I think, however, that they may be accepted as correct so far as technical results are concerned.

It is impossible to give a close and satisfactory definition of a substance which we do not fully understand, and all of the dictionary definitions of matte, as well as many of those given in metallurgical writing, seem to me to be unsatisfactory or misleading. I offer the following non-committal definition: matte is the product remaining from the sulphides of the ores after they have undergone such modification as they suffer during the process of fusion.

While copper is not an essential constituent of matte, it will, of course, always be present when copper ores form part of the furnace-mixture, which is supposed to be the case in the present remarks. Hitherto I have assumed that matte consists of varying proportions of cuprous and ferrous sulphides ($Cu_2S + xFeS$), with which is usually associated a certain amount of Fe, resulting from the dissociation of FeS by heat alone at high temperatures. So pure a matte as this, however, would be a rarity, because the ores from which the matte originated commonly contain small amounts of other metallic sulphides, which enter the matte to a greater or less extent, and must be reckoned with in succeeding operations. Such silver as may be present is usually dissolved in the matte as a sulphide (Ag_2S), while gold is present in metallic form, apparently dissolved in the sulphides. A certain proportion of such arsenic, antimony, tellurium, and selenium as the ores might have contained will also be dissolved in the matte (as arsenides, antimonides, etc., of metals), and will make trouble through all the succeeding processes, because they have a peculiarly injurious effect upon the quality of the copper.

Regarding matte as a substance to be refined — with metallic copper as its end-product — it may be said, in a general way, that the fundamental principle upon which the various stages of the operation rests is identical with the great principle which underlies nearly all the fire-refining processes by which metals are purified: namely, that the foreign substances have a greater affinity for oxygen than has the metal which we desire to produce. Some of the impurities, when thus combined with oxygen, are volatile and pass away in the gases; the others, when thus combined with oxygen, are solid or liquid, and, when properly fluxed, form a slag which floats upon the heavier metal, and thus pro-

motes an easy separation. Some small portion of the impurities always remains behind with the copper, and cannot be completely oxidized without carrying the process so far as to oxidize an undue amount of the copper itself. Consequently, the separation of foreign substances from matte (or even metallic copper) is never an absolutely perfect operation; and the quality of the final copper-product depends upon whether the small fraction of impurities which still remain are of a kind to damage the nature of the copper sufficiently to unfit it for industrial purposes, or whether they happen to be comparatively harmless. In the latter case they are disregarded; in the former the metallic copper, which has resulted from our oxidizing operations upon the matte, is dissolved in acid, and is precipitated by the aid of the electric current in the shape of exquisitely pure copper, which requires only a single rapid melting to produce it in the shapes required by the market. This electrolytic operation belongs with the refining of copper, and does not concern us in the present chapter, which busies itself solely with the production of impure metallic copper from matte.

In beginning the study of the treatment of matte, it will be more convenient to neglect all the impurities excepting iron and sulphur, and return temporarily to our original conception of matte as a mixture of cuprous and ferrous sulphides. We have, then, two impurities to eliminate, and we are told that they are to be eliminated by processes involving oxidation. This simplifies the problem, as the oxidizable product of sulphur is gaseous (SO_2), and will remove itself from the furnace without trouble, leaving us only the iron to deal with.

The removal of oxidized iron by furnace methods is already a familiar subject. We know that there are only three oxides of iron which can form under our conditions, and of these, the two higher oxides are out of the question, as they form a sticky, half-melted, thick slag. We must, then, remove our iron as FeO; and, as FeO cannot exist except in combination, we must provide SiO_2 for it to combine with and form a fusible slag. This establishes the principles of the operation by which we intend to remove the iron and sulphur from the copper, and it is now reduced to a matter of details.

What method has experience taught to be the best for burning off the sulphur of the matte, slagging its iron content, and leaving the metallic copper behind?

It is impossible to reply to this question categorically, because the answer is modified greatly by the proportion of copper that the original matte contains. The processes in use for the production of metallic copper from matte do not begin to show their individuality, or their peculiar advantages, until the percentage of copper in the matte becomes sufficient to prevent its being completely overshadowed by the iron and sulphur. I will place this boundary line, arbitrarily, at 35 per cent copper, because the matte resulting from the first smelting of copper ores is usually as rich as this, and generally several per cent richer. Mattes which contain less than this proportion of copper I will regard as sulphide ores, and will speak of their treatment in that connection. It is not until they have been concentrated up to 35 per cent of copper, or higher, that they will find place in this chapter.

Experience teaches us that there are only two methods for the production of metallic copper from matte which are worthy of serious consideration in the present condition of the art:

(*a*) The Welsh blister process which, as it can only operate advantageously upon matte consisting mainly of cuprous sulphide, must be preceded by one or more roastings and smeltings, if the matte is not already highly concentrated.

(*b*) The converter process, by which the matte is blown up to metallic copper in a modified bessemer converter. This is infinitely the more important method, having displaced the old blister process almost entirely in America, and very largely in all important copper-producing countries.

(*a*) *The blister process.*—As the chemical reactions upon which this beautiful process is based proceed comparatively slowly, and are greatly hampered by the presence of slag, it is not advantageous to employ it on mattes containing over 10 per cent of ferrous sulphide which, after allowing for a few per cent of foreign sulphides, would correspond to about 70 per cent copper. Indeed, it is worth while to go to considerable trouble to concentrate the matte to 75 per cent copper, or more, before sending it to the blister furnace.

As we are considering the treatment of any matte down to 35 per cent of copper, and as no smelter would produce, at his first fusion of the ore, a matte rich enough for the blister process, it follows that this must be preceded by one or more concentration processes, by which enough of the iron and sulphur may be re-

moved to bring the proportion of copper up to the required standard. The customary way to effect this concentration is to crush (or granulate) the low-grade matte, roast it in ordinary roasting furnaces until a certain proportion of its sulphur has been removed and its corresponding iron oxidized, and then smelt it with sufficient silica to flux the oxidized iron. This slag is usually too rich in copper to be discarded, and is added to the ore furnaces.

This concentration-smelting of the roasted matte may, naturally, be effected either in the blast-furnace or in the reverberatory. The Germans have always preferred the blast-furnace, and, in order to retain their roasted matte in the lump form so essential to this apparatus, used to roast it in heaps or stalls, repeating the roasting half a dozen times, or more, before sufficient sulphur was removed to yield a matte of the desired grade when smelted. The increased wages and the trend of modern ideas have pretty much eliminated this slow and laborious system, and the English reverberatory blister process has replaced it in the comparatively few cases where converters are yet lacking.

The English, with cheap long-flamed coal, and abundant fire-clay and fire-sand, prefer the reverberatory for all copper-smelting processes, excepting for the re-smelting of slags.

The former matte-concentration processes, both in Germany and in England, have become greatly modified by the introduction of electrolytic refining. Formerly, the quality of the refined copper was secured by a considerable number of alternate roastings and fusions whereby the matte was advanced only a little each time, and the elimination of arsenic, antimony, and other fatal impurities was more complete than would be the case if the matte received a single thorough roasting, and yielded a highly concentrated matte at a single fusion. The electrolytic process now removes the impurities which formerly had to be removed by all these extra roastings and fusions, and it is now customary to roast the original ore-matte in such a manner that a single melting will produce the highly-concentrated matte which is desired for the blister process. This practice has simplified the old English and German systems of copper smelting to an extent that can scarcely be appreciated by one who has not been personally familiar with the long series of different mattes, rich slags, and by-products to which it gave rise.

The roasting of the crushed low-grade matte differs little from

the roasting of ore, excepting that matte — being more fusible than most ores — cannot be subjected to a high temperature until enough of its grains have been oxidized to lessen its tendency to melt together and form lumps. It is roasted without difficulty in mechanical furnaces, as soon as the attendants have learned its peculiarities.

Its smelting also is simple, the main precaution being to provide it with sufficient quartz (ores, if possible) to prevent its eating away the fettling too rapidly, and to keep the hearth well filled with matte to avoid its rapid destruction by the oxides of the charge. A hearth of silica brick is greatly preferable to the ordinary sand bottom, which soaks up a vast quantity of matte, and is harder to keep in condition. The high tenor in copper of the matte produced by this smelting removes any tendency for it to cut through sides or bottom and, with a brick hearth and well-fettled sides and bridge, the smelting of roasted matte is an easy and comfortable operation. The slag, of course, will be too rich to throw away, and will go to the ore blast-furnaces, costing little or nothing to smelt, as its loosening effect upon the charge and its capacity for still taking up a few per cent of silica make it a welcome addition.

Assuming that we now have our highly-concentrated matte, we may begin the study of the blister process, one of the most beautiful of all metallurgical operations, although soon to be an object of curiosity to students of metallurgy.

The blister process derives its name from the flattened protuberances, or blisters, that cover the surface of the pigs of coarse copper, or blister copper, which constitute its final product. These blisters result from the escape, on cooling, of sulphur dioxide gas which was held in solution in the bath.

A bath of melted matte, especially if it be low in copper and high in iron sulphide, when subjected to the passage of a current of air over its surface, oxidizes with such extreme slowness as to debar this simple method of procedure in actual work. Even rich mattes, consisting chiefly of Cu_2S, undergo but a slow concentration in this manner, and the blister process would never have attained its importance had the smelter simply melted down his white metal rapidly, skimmed it clean, and then waited for the air-current over its surface to complete the required oxidation.

The really important agent in "roasting for blister" is the familiar series of reactions between oxides of copper and cuprous sulphide, especially $2Cu_2O + Cu_2S = 6Cu + SO_2$. As this indispensable reaction depends upon the presence of abundant Cu_2O the first object of study in the blister process will be to see in what manner the smelter proceeds to convert a portion of his cuprous sulphide into cuprous oxide, in order that the latter may react upon still other cuprous sulphide.[1]

The rich matte which is to be the object of this treatment is charged onto the hearth of the blister furnace, or "roaster," in the shape of large pigs weighing 200 to 300 pounds. (One pound = 0.4536 kg.). These pigs usually result from a concentration of poorer matte in the reverberatory furnace, and should contain from 70 to 80 per cent copper for satisfactory work. The amount of harmful impurities that they contain depends upon the composition of the ores from which they resulted, as well as upon the number of alternate roastings and smeltings which have intervened between the original ore and the present product. I shall speak later of the elimination of impurities in reverberatory smelting, and will say only that if the injurious substances — arsenic, antimony, tellurium, selenium, and bismuth — were present in the ores in any appreciable amount, there will still be a serious quantity of them left in the "white metal," and sufficient of them will persist in accompanying the blister copper to affect its quality. As will be seen in a moment, their elimination during the blister process may be aided remarkably by interrupting the operation to make a small amount of metallic "bottoms," which will remove a great part of the worst impurities, as well as almost all of the gold that may be present.

The heavy pigs of white metal being introduced through the side-door of the blister furnace by means of a long paddle, the furnace is closed, and the fire is urged sufficiently to bring about a softening and, finally, an extremely slow fusion of the pigs of matte. An ample supply of air is admitted through rear ports — sometimes preheated by passing it through the channeled bridge-wall — and, as the melted matte drips slowly from the pigs, it exposes a large surface to oxidizing influences. A good

[1] In the nomenclature of the English smelter, the term "roasting" is applied to this slow oxidizing fusion of large pigs of matte in a certain form of reverberatory smelting furnace known as a "roaster."

deal of the S is burned to SO_2, and some of the copper with which it was combined is oxidized to Cu_2O and even to CuO.

This extremely slow oxidizing fusion is continued until the pigs have all flattened out into a pasty mass, and the exposed surface of the latter has become so blanketed with oxides that little more would be gained by continuing this stage. The temperature is now raised materially, and the second period of the "roasting" is inaugurated. This may be called the reaction period, the first stage being distinctively the oxidizing period. The oxides and sulphides that are in contact react to form metallic copper and SO_2 gas, as already described; the small amount of iron present is oxidized and slagged, finding silica in the sand adhering to the pigs of white metal, and in the fettling of the hearth; a moderate proportion of the arsenic and antimony is removed (mainly by scorification), and a most unsatisfactory fraction of the bismuth (mainly by volatilization). The less injurious metallic impurities are slagged to a considerable extent.

During this reaction period a large amount of copper is bereft of its sulphur, and forms a metallic bath upon which the remaining undecomposed matte floats. The gradual cessation of the bubbles of gas indicates that the time of usefulness of this stage is drawing to a close, and arrangements are now made by the furnace-man to reproduce the conditions of the first period so far as is practicable. The slag is skimmed clean, the fire-door is opened, and the thoroughly liquid charge once more falls back into the pasty condition of semi-fusion. As the evolution of SO_2 gas still continues to some extent, the bubbles force their way up through the pasty mass, covering it with little craters and excrescences, and multiplying the area exposed to the action of the air. This is continued as long as active oxidation progresses, when the heat is raised and the second reaction period is inaugurated. The cuprous oxide is dissolved rapidly in the bath of metallic copper, acting vigorously as a carrier of oxygen to cuprous sulphide, as well as to foreign sulphides.

Finally, in from 18 to 30 hours from the beginning of the process, the sulphides have disappeared, a strong heat is given to slag all metal oxides, the face of the bath is skimmed clean, and the "coarse copper" (blister copper) is tapped into sand moulds. The pigs are broken apart with bars as soon as they have solidified sufficiently to allow the necessary prying, as the metal is weak and

brittle ("red-short") when hot, but could not be broken if allowed to cool.

One of the serious drawbacks of the blister process is the large amount of rich slag produced, which adds materially to the cost of the operation. Another disadvantage is the tying-up of exceedingly large quantities of metallic copper in the sand bottom of the hearth. This seems to be an essential feature of the operation, as it is not until the sand is pretty thoroughly soaked and cemented with metallic copper that it is firm enough to withstand the boiling of a bath of from 6 to 20 tons, or more, of blister copper. Brick bottoms have been used for this purpose with entire satisfaction, provided they are guarded against injury when charging the heavy pigs of white metal. This is sometimes accomplished by tapping the blister at such a level that a sufficient portion remains permanently upon the hearth and, when chilled, protects it from the mechanical injuries likely to occur during charging.

Allan Gibb, of Queensland, Australia, has paid much attention to the practical results obtained in the elimination of impurities from copper in the reverberatory furnace, and gives many most valuable observations upon this subject, to which I shall refer in another place. Meanwhile, I append a table from one of his monographs, which shows very distinctly the extent of the elimination of arsenic, antimony, and bismuth during the blister process, and the manner in which this elimination occurs. It will, of course, be understood that each separate case is a law unto itself, and that the extent and manner of elimination are influenced by many factors.[1]

BLISTER-ROASTING OF WHITE METAL
ELIMINATION OF ARSENIC, ANTIMONY, AND BISMUTH

	Weight Tons	Arsenic		Per cent relative original arsenic = 100
		p. c.	Tons	
White metal roasted	100.0	0.089	0.089	100.0
Products	—	—	—	—
Blister-copper	70.1	0.083	0.058	65.2
Roaster-slag	16.3	0.092	0.015	16.8
Total			0.073	82.0
Difference due to volatilization			0.016	18.0

[1] "The Elimination of Arsenic, Antimony, and Bismuth from Copper," by Allan Gibb; *Trans. Am. Inst. Mng. Engrs.*, Vol. XXXIII, p. 653.

	Weight Tons	Antimony		Per cent relative original antimony = 100
		p. c.	Tons	
White metal roasted	100.0	0.039	0.039	100.0
Products	—	—	—	—
Blister copper	70.1	0.024	0.017	43.6
Roaster-slag	16.3	0.084	0.014	35.9
Total			0.031	79.5
Difference due to volatilization			0.008	20.5

	Weight Tons	Bismuth		Per cent relative original Bismuth = 100
		p. c.	Tons	
White metal roasted	100.0	0.015	0.015	100.0
Products	—	—	—	—
Blister copper	70.1	0.003	0.002	13.3
Roaster-slag	16.3	0.012	0.002	13.3
Total			0.004	26.6
Difference due to volatilization			0.011	73.4

It has been already noted that certain of the impurities that are contained in the pigs of white metal subjected to the slow roasting-fusion of the blister process have so much greater an affinity for the metallic copper than they have for the matte in which they are dissolved that they leave the matte and become concentrated in the metallic copper as it forms.

The Welsh smelters have long taken advantage of this fact to purify their matte and concentrate the greater portion of the most objectionable impurities in a small amount of metallic copper, thus obtaining a residual matte much more free from arsenic, antimony, bismuth, etc., than was the original white metal. This matte is worked separately for a high-grade copper, called "best selected," and the small proportion of metallic copper containing the impurities, known under the name of "bottoms," undergoes special treatment.

The importance of the bottoms process is greatly enhanced by the fact that gold also has a remarkable affinity for metallic copper, as compared with matte, so that, under suitable conditions, almost the entire gold content of a large quantity of matte can be extracted in these impure bottoms. This fact was pecu-

liarly important before the perfection of the electrolytic process for the refining of copper, as it permitted the profitable extraction of small quantities of gold from mattes which did not contain a sufficient amount of that metal to be submitted to the expense of treatment for it.

While the electrolytic process has rendered this older method superfluous in most smelting enterprises, it is still used sufficiently to demand a brief consideration, and will no doubt continue to be employed, under exceptional conditions, for years to come. The most important feature of this interesting process, to the modern smelter, is the facility with which the gold content of the matte may be concentrated in a comparatively small amount of metallic bottoms.

The first point that the practical metallurgist would desire to establish would be to learn just what proportion of the Cu_2S of his matte he must convert into metallic copper in order to ensure a satisfactory separation of the gold. This point has been studied extensively by Allan Gibb, from whose work,[1] reliable information upon the subject may be obtained, although it must always be borne in mind that exceedingly minute quantities of other impurities may have a profound effect upon the amount of metallic copper which may be required to extract gold from the matte with the thoroughness required. It must also be recollected that the mattes studied by Gibb consisted for the most part of nearly pure cuprous sulphide, in which substance metallic copper is practically insoluble, and therefore separates with great ease. This is, however, the condition which would be present in the ordinary production of bottoms for commercial purposes, and thus is applicable to the present subject.

Gibb found that "with a reduction of 8.2 per cent of the total copper (of the matte) to bottoms, the latter contained 41.5 per cent of the total gold; whilst, when 14.4 per cent of the copper is reduced to bottoms, the whole of the gold is found in them. In some cases traces of gold remained in the regulus (the rich matte which still remains); but they were too small to be weighed, even when 50 grams was taken for assay."

This indicates, therefore, that quantity plays a very important

[1] "The Elimination of Impurities during the Process of making Best Selected Copper." Third report of Alloys Research Committee, Institution of Mechanical Engineers, p. 254 (1895).

part in the operation and that, in a comparatively pure Cu_2S matte, something near 14 per cent of its total copper must be reduced to bottoms in order to obtain a complete extraction of the gold.

In some experimental work executed in 1884 on Butte mattes which had been enriched in gold by adding rich gold ores to the copper-charge in the ore-furnaces, and which contained about five ounces gold per ton of white metal (0.017143 p. c.), I began with reducing about 10 per cent of the copper in the matte to metallic copper. The gold extraction being found complete, the proportion of bottoms was gradually reduced to about 6 per cent, at which point the extraction was sufficiently good for commercial purposes, the depleted matte assaying from 0.05 to 0.08 oz. gold per ton (0.00017 to 0.000274 p.c.). It must be recollected, however, that these Butte mattes contained small quantities of various impurities, which appear to aid in the collection of the gold and thus permit its extraction with a much smaller proportion of bottoms than would be the case with a pure matte.

The most available and familiar object-lesson (to Americans) is the bottoms process as introduced by Richard Pearce at the Boston and Colorado smelter in Colorado, and practised for some 35 years, producing — in connection with Pearce's process for the recovery of the gold contents of the bottoms — over 1,600,000 oz. (49,767 kg.) of fine gold.

In his description of the process Pearce says[1] (referring to matte containing 65 per cent copper, with 10 to 15 oz. gold and 80 oz. silver per ton):[2]

"Twelve tons of the matte are placed in slabs on the hearth of a large reverberatory furnace. The openings of the furnace are closed except two large ports near the bridge, which are used to supply the air for oxidation. The heat is gradually increased for a period of about seven hours, or until the whole mass assumes a semi-pasty condition. The ports are then closed, and the heat is still further increased until a point is reached when the whole charge is in a perfectly molten condition. In the last stage of the melting, a reaction takes place between the oxides and sulphides, and a certain amount of metallic copper is liberated. The metallic copper contains nearly all the gold, together with any foreign

[1] *Trans. Am. Inst. Mng. Engrs.*, Vol. XVIII, p. 68.

[2] One ounce per ton = 0.0034286 p. c.

elements present in the matte, such as lead, arsenic, antimony, bismuth, etc. The charge is tapped into sand moulds, in the first few of which, nearest the tap-hole, there will be found, after cooling under the enriched matte, plates of the above-mentioned impure copper, commonly known as metallic bottoms. The proportion of these bottoms to the whole tap of matte, called *pimple metal*, is about 1 to 15. The following analysis will give some idea of the composition of this crude alloy:

copper	60.04	per	cent
lead	33.61	"	"
arsenic	0.44	"	"
bismuth	0.40	"	"
iron	0.08	"	"
zinc	0.15	"	"
sulphur	1.68	"	"
gold	0.54	"	"
silver	1.35	"	"
	98.29	"	"

"It must be understood that the analysis shows the composition of one specimen of such alloy only. The composition varies, of course, with the nature and quantity of the impurities present in the residue metal. These impurities, without a doubt, increase the solvent action of the copper on the gold. In other words, a copper matte containing little or none of the impurities mentioned, will not yield its gold so readily to the copper. The whole operation of roasting and smelting requires about twelve hours. The crude bottoms contain 100 to 200 oz. gold per ton, and about 300 oz. silver. The residual matte (pimple metal), if the operation has been properly done, contains not more than 0.2 oz. gold per ton (frequently less than 0.1 oz.), 90 oz. silver, and 77 per cent copper. The roaster-slag from this operation is smelted in the ore furnaces. The treatment of the crude alloy, and the separation of the gold from the copper, are a part of the process which, for certain business considerations, I am not at liberty to describe."

Since this description, the business considerations which prevented Mr. Pearce from describing the interesting method which he originated for the treatment of these gold-copper bottoms have been removed, and his son, Harold V. Pearce, has described this

operation in a most interesting paper, having the benefit of his father's notes and aid in preparing the description.[1]

While wet methods, either for the extraction of copper from its ores or for the recovery of the precious metals from matte, can find no place in this work, strictly furnace-operations are in harmony with its general scope, and I feel confident that a brief description of this unique gold-separation process may be of great use under certain conditions which any copper metallurgist is likely to encounter. For instance, imagine a remote plant shipping rich matte to a seaboard refinery. The ores are so pure that the copper will not have to be submitted to the electrolytic process of refining. They are also low in silver, but contain a moderate amount of gold, insufficient in quantity to warrant submitting the entire copper product to electrolysis, and yet too large to see wasted in the ingot copper. The natural solution would be to make a small amount of bottoms in a reverberatory furnace, which would thus concentrate all of the gold, and a fair proportion of the silver, in a small amount of metallic copper. The question would at once arise: What is to be done with these gold-copper bottoms? Not only is the refiner's tariff for their treatment exceedingly high, but the adjustment of assay between buyer and seller on rich gold-bearing products of this nature is most harassing and unsatisfactory to the seller. The employment of the Pearce process offers a solution of the difficulty, and is quite within the powers of any careful chemist or assayer.

The following description is taken mainly from the paper above mentioned:

The Pearce gold-separation process may be regarded, to a considerable degree, as the logical extension of the preceding bottoms-process. That is to say, it depends upon the fact that gold will follow the metallic copper rather than the matte, and also that copper has a great affinity for sulphur.

Applying these principles, we find that a granulated copper-gold alloy, melted with a suitable quantity of pyrite (FeS_2) will form a certain amount of matte — some of the copper combining with the excess sulphur of the pyrite and mixing with the remaining FeS — while the rest of the copper will still remain in the metallic form. This residual copper will contain most of the gold

[1] "The Pearce Gold-separation Process," by Harold V. Pearce; *Trans. Am. Inst. Mng. Engrs.*, Vol. XXXIX, p. 722.

present and will thus be much richer in that metal than was the original copper-gold alloy. Pearce originated a sequence of operations by means of which this principle could be applied commercially. It remains for me only to give some slight sketch of this series of concentrations — called by Pearce "strippings," because a certain proportion of the copper is stripped from the gold at each operation, until the latter has become rich enough for a final refining with nitre, and the production of an ingot of sufficient purity to be accepted by the U. S. Mint — 900 fine. Each stripping must be preceded by a granulation of the alloy in order that the pyrite may have proper opportunity to react therewith.

Beginning the process with the bottoms already described and containing 0.4 to 0.5 per cent of gold (say 125 to 150 oz. gold per ton of bottoms), it requires usually five strippings with corresponding refinings and granulations, the first melting being conducted in a small reverberatory furnace, whilst the last operations are executed in plumbago crucibles. The occasion for this long series of operations results from the fact that any attempt to remove too much of the copper at a single stripping will result in making the accompanying matte too rich in gold. The material under treatment is so enormously rich, and represents so high a degree of concentration, that, in spite of its apparent complexity, its cost — when calculated back upon a ton of original ore — becomes trifling.

From Harold Pearce's paper I abstract the following description of the practical working of the process:

1. Treatment of the impure gold-bearing copper bottoms, containing about one-half of one per cent of gold. — A charge of 6400 lb. of the exceedingly impure copper-gold bottoms is placed upon the brick hearth of a small reverberatory furnace, and gradually melted in an oxidizing atmosphere. When sufficiently refined, it is skimmed clean and tapped into a tank of cold water, the stream of copper falling upon a wooden pole, to spray it before it reaches the water. This is the *first granulation.*

2. First stripping. — Twenty-two hundred pounds of the above granulated alloy is mixed with 1500 lb. of clean pyrite, one pound of pyrite being expected to remove about one pound of copper. The mixture is charged into a second small reverberatory furnace for the first stripping. When skimmed, the charge is tapped into a series of sand-moulds, the matte having a reddish color and a

composition analogous to the mineral bornite. It is called red metal. Underneath the first three pigs of matte will be found copper bottoms, aggregating 625 to 650 lb. in weight. These will contain two to three per cent of gold.

3. Second granulation. — These enriched bottoms — after several tons have accumulated — are now charged back into the granulating furnace, and the method described in No. 1 is repeated, so that granules are again obtained for the next step.

4. Second stripping. — Twenty-two hundred pounds of these granules are melted with a corresponding amount of clean pyrite for still richer bottoms.

5. Third granulations. — Melting and granulating of the rich bottoms from No. 3. Great care is taken with this third granulation, in order to obtain as pure a product as possible for the crucible stripping and refining operations which succeed it. About 12 hours is employed in a slow oxidizing fusion before the alloy is pure enough to tap and granulate.

These three granulations and two strippings complete the portion of the work which is carried out in small reverberatory furnaces, for the product of the third granulation is now so rich that it is treated henceforth in large plumbago crucibles.

The granulated alloy from No. 5 now contains from 12 to 18 per cent of gold. It is mixed with about one-half its weight of clean pyrite, and melted with borax in plumbago crucibles, which will hold about 40 lb. of the granules and its accompanying pyrite. The contents of the crucible is poured into a conical cast-iron mould, and allowed to cool. The enriched copper bottom is found at the apex of the cone, and the matte is easily broken away from it. It then undergoes refining with nitre and is again granulated by ladling it into a barrel of cold water.[1]

Two to three repetitions of this stripping in crucibles are required to produce gold bars 900 fine. The very rich matte from the final strippings is melted with a small quantity of metallic iron. This displaces a certain amount of copper, which carries with it much of the gold that was contained in the matte. The poorer mattes and the slag present no especial difficulties, and con-

[1] The skimmings during this refining contain a large quantity of bismuth. These are collected and subsequently melted in a crucible to yield a bottom consisting largely of bismuth with other impurities, and containing two to ten per cent of gold.

tain so small a proportion of the original gold that they may be worked up by ordinary methods.

The final point which is of the most importance to the practitioner is to learn what loss is incurred by this method, and what proportion of his original gold he may hope to turn out promptly in the shape of bars ready for the mint.

In manipulating such enormously valuable metallic alloys as we are now considering, it is impossible to cut out any one fraction of the year's work, and produce an accurate balance sheet of the results. Nothing but an elaborate stock-taking at long intervals can give the data required for this, and even such a radical clean-up is imperfect, owing to gold tied up in furnace hearths, partly-used crucibles, flue-dust, and in similar places. Apart from the difficulty of sampling, however, gold is a satisfactory metal to deal with, owing to its stability.

Mr. Pearce gives detailed statements, covering a great quantity of material, which indicate so small a loss of gold in the process that it can scarcely be detected, the errors in sampling and assaying burying it so completely that it would be idle to attempt to establish any arbitrary figure.

The separation of the silver in mattes in the form of metallic copper bottoms carrying silver is very imperfect as compared with gold, although sufficiently marked to show that silver has a decided affinity for the metallic copper. Its behavior is also complicated by the effect of the impurities present, which seem to influence its concentration to an even greater degree than they do in the case of gold. The subject possesses but little literature, and that of an unsatisfactory and contradictory character.

Gibb found that silver nearly reached its maximum concentration in bottoms when 19 per cent of the total copper was separated in that form. They then contained 42.9 per cent of the original silver. The presence of nickel and arsenic in large amounts seems to have an injurious effect upon the concentration of the silver, as Gibb states that when 19.3 per cent of bottoms — containing about 10 per cent each of arsenic and nickel — were formed, they contained only 26.7 per cent of the silver in the original matte.

In extracting the gold from a 65 per cent copper matte at the Boston and Colorado smelter, and reducing to (very impure)

bottoms about 6.6 per cent of the original copper (thereby extracting practically the whole of the gold), Richard Pearce found that about 50 per cent of the silver content of the matte entered the bottoms.

My own limited experience in this matter leads me to believe that — omitting exceptional cases — about 40 per cent of the silver content of the matte will enter the bottoms, provided a sufficient proportion of the latter is formed to make a good extraction of the gold.

The construction of the blister furnace presents no unusual features, excepting that provision should be made for the free admission of air during that stage of the process where oxidation plays so important a part. This is effected by leaving ports in the rear wall on either side of the bridge, or by allowing the air to circulate through the channeled bridge-wall, and enter the hearth through several small openings in its anterior wall. This preheating of the air has always seemed to me rather fanciful, as there is no lack of heat in the blister furnace, and the interior air openings become choked and demand frequent attention.

It must, of course, be heavily ironed to withstand the great quantity of melted copper which it is to contain; but we iron all of our reverberatory furnaces strongly at the present time, and expect all of them to carry a bath of matte that would have seemed ruinous twenty years ago.

The construction of the bottom is a vexed question. Personally, I feel strongly that a brick bottom is a safer foundation for the bath of boiling copper than the classic sand bottom. The brick bottom is used at many furnaces in Europe and, so far as I am aware, has never been discarded when once introduced. The sand bottom is not safe for the exacting blister work until it has been cemented into a solid metallic block for several inches in depth by the absorption of copper; and this absorption continues slowly but inexorably in depth, until the amount tied up is almost incredible. The final removal of one of these bottoms is almost like cutting up a hundred-ton mass of metallic copper at one of the Lake mines.

The brick bottom, on the other hand, absorbs so little copper that the amount is almost negligible, even on a new bottom, and even this slight drain ceases in a short time.

I have never, myself, had any trouble with a brick bottom either in ore smelting, or in refining, although I have never used the large modern furnaces.

The success of such a bottom depends chiefly upon three factors: the soundness of the foundation upon which the brick bottom is constructed; the manner in which the inverted arch, forming the bottom, is laid; the character of the brick that are used.

That there is no inherent objection to the employment of such a bottom in blister or refining furnaces is evinced in a striking manner by the furnaces of The Anaconda Co. which receive the converter copper, and give it a partial refining, prior to casting it into anode plates for shipment to the electrolytic refinery. I know of no bottoms that are subjected to such trying conditions as these. Each furnace handles an average of 300,000 lb. (136,080 kg.), or one and one-half charges, per 24 hours, and they do not enjoy the respite granted to the ordinary furnace of a slight chill for the bottom during the period of charging. At these Washoe furnaces the converters begin pouring in a fresh charge of copper as soon as the last of the preceding charge is tapped out, and the introduction of the air-blast (through pipes plunged in the molten bath) often begins before the furnace has received its full 200,000 lb. from the converters. The arch lasts only four to five months, owing largely to the constant splash of the boiling metal; but the bottoms endure at least four years of constant use. These furnaces will be described in detail in a later chapter.

(*b*) *The converter process.* — The blister process, admirable as is its chemistry and its result, was too slow for modern conditions. Moreover, its radical lack of continuity unfits it for an age which demands that its methods shall advance continuously, and that the few absolutely essential shifts of material shall be effected by mechanical contrivances and with a minimum of delay.

These demands are fulfilled by the converter process to an unusual degree. Receiving its liquid charge from the ore furnace, it blows it to metallic copper without human intervention, except in guiding and directing, and pours the finished product into the refining furnace, or casts it into anode plates suitable for electrolysis. Its one weak point is the constant repairing demanded by the lining of the converters, and this drawback may, perhaps, disappear with the perfecting of the basic-lined converter.

That the heat evolved by the oxidation of both iron and sulphur was sufficient to melt these substances is a fact that has been known, theoretically, for a long time. It was only, however, about the middle of the last century that experimenters began to recognize the fact that the heat produced by this oxidation increased rapidly under pressure and that, conversely, this higher temperature increases amazingly the velocity of the chemical reactions themselves.

As well put by Sticht,[1] "The time element is a vital factor in oxidation processes if a separation of the products by purely physical means is desired, such as differences of specific gravity, which constitute so important a feature of smelting processes in general. The necessity of concentrating the reactions involved into a minimum of time, and, it may also be added, as far as the gaseous oxidizing agent is concerned, into a minimum of space, is thus obvious. Under all circumstances, whether oxidation proceeds slowly or rapidly, only the same amount of heat is generated. But its dilution, so to say, by spreading it over long periods of time is inimical to the attainment of a pyrometric intensity reaching and surpassing the fusion points of the solid products involved."

The neglect of this vital time-element led to many disappointments in the early days of copper converting, and is still much neglected in the operation of partial pyrite smelting which is becoming so universal a process. It is the most important of all principles for the metallurgist who desires to make use of the rapid oxidation methods, and who thus intends to burn chemical fuels instead of coke.

Omitting, regretfully, the admirable working experiments of Semennikow,[2] in which he demonstrated the advantage of blowing 31 per cent matte up to white metal (failing, for want of bulk of material, to effect the successful production of metallic copper), we must pass to the work of Pierre Manhès who, in 1880, at his copper smelter at Vèdenes, France, surmounted the difficulties

[1] "Progress in Rapid Oxidation Processes applied to Copper-Smelting," presidential address of R. S. Sticht before the Australasian Association for the Advancement of Science, January, 1907. I make free use of certain of the views embodied in this address.

[2] *Russian Mining Gazette* of 1870, and *Berg-u. Hüttenmänn. Ztg.*, of 1871.

attending the production of metallic copper in the converter, and placed the bessemerizing of copper mattes on a commercial basis.

The bessemerizing of pig iron, for the production of steel, is so familiar a process, and one so highly perfected that, at first glance, it seems strange that the copper metallurgist, with such an object-lesson before him, has found such extreme difficulty in adapting its principles to the bessemerizing of matte, after it had once been established that the combustion of its iron and sulphur were capable of supplying sufficient heat to carry the process through to the production of a high-grade pig copper.

A brief comparison between the behavior of these two substances — pig iron and copper matte — when subjected to the operation of bessemerizing, will form a useful introduction to the study of the copper converter process.

Pig iron, in the converter, yields a single, homogeneous, metallic product, loses but little in bulk or in weight during the blowing, makes but little slag, and has but little destructive effect upon the lining of the vessel.

Ordinary matte containing, perhaps, 45 per cent copper, 25 per cent each of Fe and S, and 5 per cent of other impurities, forms two — and, during a portion of the time, three — distinct layers of melted products, loses nearly 60 per cent of its weight and more than that of its bulk, and has a highly corrosive and destructive effect upon the converter lining.

All of these peculiarities add materially to the difficulties experienced in adapting the converter to the treatment of copper matte, but the truly fatal obstacle to all of the earlier attempts was the fact that the small bulk of metallic copper that remained at the end of the blow was chilled by the jets of air which penetrated it from below, and, not having body enough to pour, chilled and adhered to the lining of the vessel, coating it with half an inch, or more, of metallic copper of 95 per cent and upward. The device of "doubling," and blowing several charges of matte, one after the other, without attempting to pour the resulting copper until a suitable quantity has accumulated, had not yet been attempted, and the feature which contributed most fundamentally to Manhès' success was the placing of the belt of tuyere-openings horizontally, and several inches above the bottom. This provided a quiet chamber below the turmoil of the blast, where the metallic copper could collect in a body, ready for pouring.

In addition to the saving in time and money effected by the substitution of the converter operation for the blister process, it was found that the impure mattes, which formed Manhès' supply, yielded a much higher grade copper than was afforded by the older method.

In 1884, the Parrot Silver and Copper Company of Butte, Montana, installed the Manhès converters, with the assistance of the inventor, and soon began a series of improvements which suited the process better to American conditions, and resulted in much greater capacity and economy.

Among these improvements were the systematic punching of the tuyeres during the blow; the conversion of the matte into copper at a single operation, instead of in two stages; the tilting of the vessel by machinery, instead of by hand; and many mechanical improvements, largely due to A. J. Schumacher.

After the improvements to which I have just referred, the Parrot converter had a height of 8½ ft., and a diameter of 5 ft., externally measured. When freshly lined, it took a charge of 2500 lb. of matte, rising to 9000 lb. with the corrosion of the lining. It received its charge from a remelting furnace, placed on a higher level, and was tilted at first by belting off a line-shaft; later, by an hydraulic piston. It had 16 tuyeres, and the lining averaged a duty of 16 blows on 45 per cent matte. The shell was divided into three — later, two — sections, which facilitated the re-lining.

Figs. 88 and 89 illustrate the original Parrot converter after it had assumed its improved form.[1]

A rapid increase in the size of the vessels was a natural concomitant of their remarkable success. The Boston & Montana Co., at their Great Falls plant, built converters 7 by 14½ ft. holding 10,000 to 22,000 lb. (4536 to 9980 kg.) of matte, according to the condition of the lining. At the Aguas Calientes smelter, in Mexico, the size of the vessels was increased to 8 by 16 ft. The large upright converters in use at Great Falls in 1910 are 12 ft. in diameter.

The original type of converter used by Manhès was of the recumbent or horizontal form, resembling a barrel laid upon its side. Of late years, the tendency of the profession has been to revert to this older construction, and the majority of the plants

[1] One foot (12 inches) = 0.305 m.

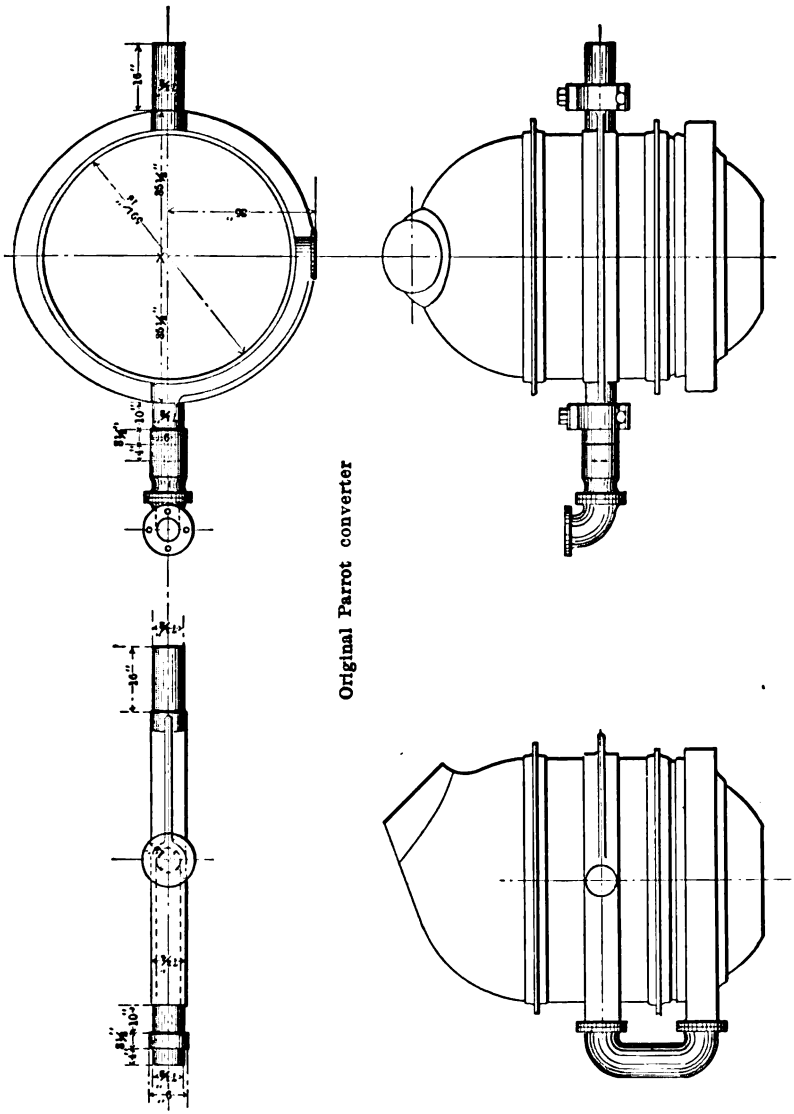


FIG. 88. — Old Parrot converter

now in operation employ this horizontal, or trough, or Leghorn converter, all of which names are applied to it.

One of the main advantages of the barrel converter is supposed to be the fact that it can operate with a lighter blast-pressure, although this somewhat feeble reason certainly does not obtain with any of the great copper companies who use it, and

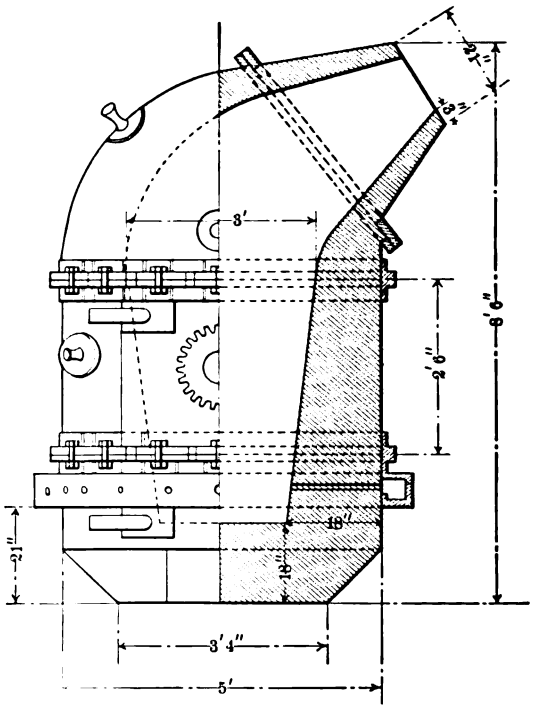


FIG. 89. — Parrot converter

who would gladly furnish a blast of any required pressure if it conduced to convenience or economy.

The reason why the barrel converter may be run with a peculiarly low blast-pressure will become evident on an examination of Figs. 90 to 94, showing the general features of this apparatus.[1]

The converter-shell here portrayed as typical of a modern barrel converter is 8 ft. in diameter, and 11 ft. long in the center.

[1] These illustrations are taken from a paper entitled "Modern Type of the Barrel Copper Converter," by Charles F. Shelby, *Eng. and Mng. Journal*, Oct. 23, 1909.

It is made of steel plate, and has cast-steel heads, on which are cast the riding-rings that rest upon the grooved supporting wheels upon which it revolves. The sectional views, Figs. 93 and 94, show how the shell is lined, and indicate the position of the row of tuyeres, which are only on one side of the shell. It will thus be evident that the vessel may be tilted during the blowing, as the bath of matte diminishes, in such a manner that the tuyeres may constantly be submerged to the same uniform depth. Thus, it is not necessary, as in the upright converter, to begin with a deep layer of matte, requiring a heavy pressure of wind.

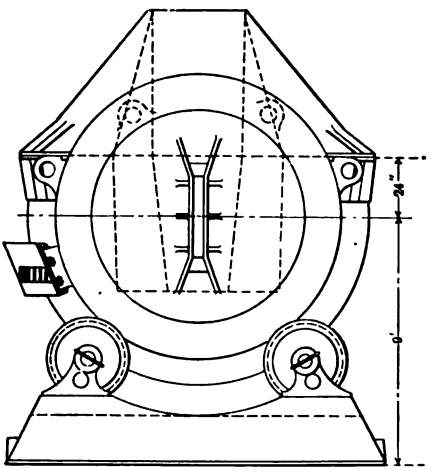


FIG. 90. — Barrel converter — end elevation

This is a valuable feature for small plants with weak blowing capacity, but does not apply to the great establishments, where they often run the barrel converter with as strong a blast and as deep a layer of matte as would be employed with the upright vessel.

The one aim and object in bessemerizing copper is not to save blast, but to oxidize sulphur and iron as rapidly as possible. If this purpose can be effected as rapidly by blowing air at light pressures through a shallow layer of matte as by powerful pressure through a deep layer, the light blast will be, of course, the one to select. But both theory and practice controvert any such assumption. No one is better equipped than the intelligent converter

foreman to pass judgment on the practical side of this question; and no one clamors more persistently for more air and a deeper column of matte when he wants to turn out copper rapidly.

This same horizontal shape is also badly suited to afford support to its lining, and is not so convenient to re-line as the upright vessel. On the whole, I am unable to see why the tide sets so strongly in favor of the barrel-shaped converter, and I find that the old established plants who *retain* the upright type — such as the Great Falls and the Mount Lyell smelters — feel strongly that the latter is the more advantageous form.

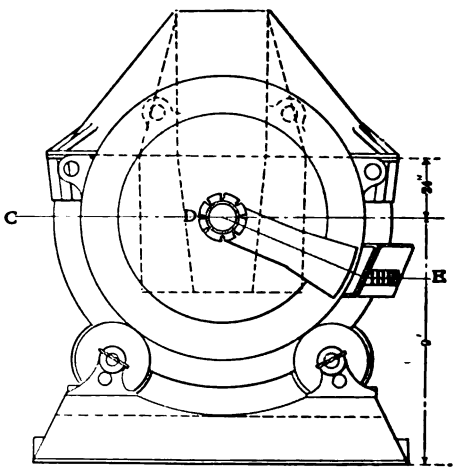


FIG. 91. — Barrel converter — end elevation

It seems probable, indeed, that bottom-blown upright vessels may, after all, be found the most advantageous, and that Manhès' device of horizontal tuyeres, with a quiet reservoir below them for the copper, may pass away now that we have learned to work with such large quantities of material that the copper needs no such protection.[1] The well-known Boston and Montana experiments, in a large upright converter with sixteen $\frac{3}{4}$ in. tuyeres (0.018 mm), brought matte to blister decidedly more quickly, and with less damage to the lining, than the ordinary vessels. The main

[1] In present practice, the metallic copper often stands four, and even six, inches above the tuyere level.

difficulty lies in making the bottom-tuyeres stand, and still later experiments indicate that this is likely to be overcome.

As the lining material of the vessel plays the double rôle of protector to the exterior shell, and flux for the process itself, it would seem logical to study the operation of converting as applied to copper mattes before proceeding further with details of construction which are modified to suit this operation.

Referring to the introductory remarks of this chapter, we are reminded that the sole object of submitting a copper matte to the process of bessemerizing is to give its sulphur and iron (incidentally, such other impurities as it may contain) opportunity to

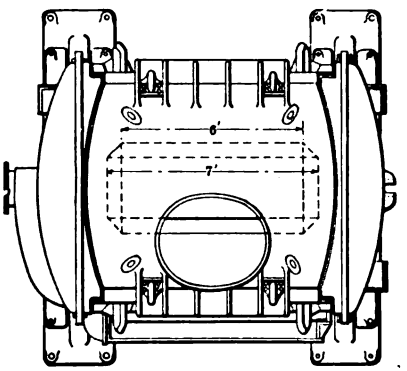


FIG. 92. — Barrel converter — top view

combine with oxygen and separate themselves, physically, from the metallic copper with which they were previously united.

Naturally, if we desire to burn sulphides by forcing air through their mass, we are obliged to have them in a liquid condition to start with; and, as it is not usually convenient to melt them in the converter, it is customary to melt the matte elsewhere, and pour it into the converter. The reactions begin, of course, the moment the converter is turned back into position, and the blast penetrates through its liquid contents.

It is customary to divide the converting operation into at least two stages: the first occupying the period from the commencement of the blowing until the ferrous sulphide of the matte is oxidized; the second, from the close of the first period until the remaining cuprous sulphide is decomposed and the metallic copper is ready to pour.

The first period is, of course, the slag-forming period, because it witnesses the oxidation of the ferrous sulphide, and the slagging of its iron with free silica, which must be provided for the purpose. The sulphur content of the FeS is burned to SO_2, and escapes as gas; the iron with which it is united forms FeO and combines with silica to form slag. In exactly what manner the Fe comports itself after its atom of S has been burned away from it, and before it meets its required SiO_2 to be slagged as FeO, we do not know. It is sufficient for present purposes to confine ourselves mainly to the final results. It will be noted that during this first, or slag-

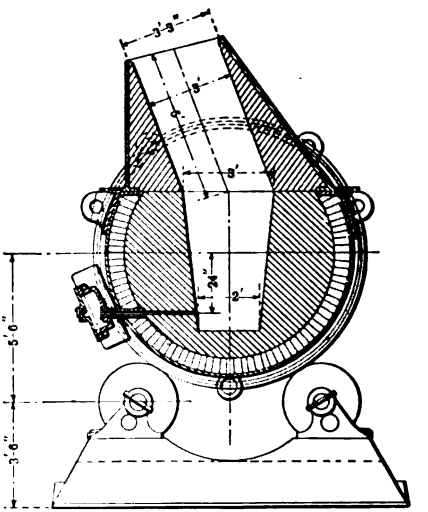


FIG. 93. — Barrel converter — vertical section

forming, period, we have three sources of heat: the combustion of the sulphur, the oxidation of the iron, and the uniting of the ferrous oxide with silica to form a ferrous silicate.

The considerable amount of slag formed during this first period is poured off, and the second period begins. The Cu_2S, being the only remaining substance, now begins to undergo rapid decomposition, the S burning to SO_2 gas, and each pound of sulphur, as it is removed, dropping the four pounds of metallic copper with which it was combined. Copper also is constantly undergoing oxidation, but is as constantly reduced to the metallic form by the accompanying cuprous sulphide ($2Cu_2O + Cu_2S = 6Cu + SO_2$).

During this second period there is but one source of heat — the oxidation of the sulphur; small as this seems, it is ample to carry the process to its close, provided the converter was at its proper temperature at the end of the first period, and that the reactions proceed with due rapidity.

The phenomena of the entire operation will be examined more in detail in succeeding pages, but sufficient of its chemistry is now understood to facilitate the study of the construction and lining of the vessel.

Despite the great affinity which exists between silica and ferrous oxide, when both are in a highly heated condition, it is a curious fact that the principal obstacle to great improvement and cheapening of the converter process is the difficulty experienced in getting these two substances to unite in any manner which shall be convenient and suitable.

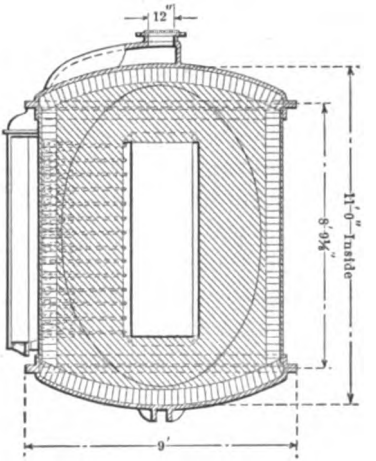


FIG. 94. — Barrel converter — longitudinal section

Without encumbering these pages with an account of the efforts that have been expended in trying to provide the FeO with some form of silica more convenient than that obtained by the eating away of the lining, I will merely say that many plans have been tried by many skilled metallurgists. Quartz sand blown through the tuyeres refused to combine with the oxidized iron and, mixing with the matte, made a sad mess that threatened to freeze solid. Quartzose material placed upon the surface of the bath floated around in sticky, agglomerated masses, but effected but little toward slagging the iron, as the action of the FeO at the surface seemed but trifling, and the lining was destroyed almost as rapidly as before. A systematic attempt to force the FeO to combine with the added quartz by lining the converter with magnesite brick, or similar refractory material, resulted in the peroxidation of the iron content of the matte. Ferric and magnetic oxides

were formed in great abundance, producing a sticky, half-fused slag which mixed mechanically with the residual matte, and had to be put through the smelting furnace again before its constituents could be disentangled.[1]

Assuming then, temporarily, that we are forced to derive our silica from the lining of the vessel, we will consider the class of material which experience has shown to be most suitable for this two-fold purpose.

Our two chief requirements are that it shall be refractory and that it shall contain the greatest possible proportion of free silica. Such silica as is already in combination with other bases is not available as a flux for the FeO. (See page 220.) There is a third requirement which, commercially, is nearly as important as the two already mentioned: it must be cheap. In addition to these absolute requirements, it would be exceedingly advantageous if this refractory, silicious material contained metal values, as they would be recovered in the matte as the lining was gradually eaten away; also, it would be well for it to be sufficiently plastic and sticky to hold together firmly when tamped into the converter shell.

It need scarcely be said that such a combination of ideal qualities cannot be expected in practice. The nearest approach to them with which I am acquainted was Hixon's Pachuca silver ore at the Aguas Calientes smelter, which seemed to possess all of these advantages, and enabled him to treat a 30 per cent matte in the converter, and thus throw upon this operation much of the work that usually belongs to the smelting furnace.

Formerly it was the practice to use crushed quartz (often quartzite or sandstone) and incorporate with it sufficient cheap, sticky clay to hold the mass together. One of the important economies of modern times is the substitution of quartzose ores for the barren quartz, and of sticky concentrator slimes — or other adhesive material containing copper — for the barren clay. The sulphides and other fusible constituents of the ore diminish the life of the lining; but, by this method, a class of ore which

[1] In recent times much greater success has attended the attempts to force the FeO of the matte to content itself with added quartz, and there seems good reason to believe that the time has come when we may no longer have to submit to the destruction of our linings to furnish flux for the iron of the matte. I shall refer later to these recent advances, but am speaking at present of the ordinary conditions that will be found in most of the copper bessemerizing plants of the world.

is costly to smelt, or wasteful to concentrate, is treated for nothing. As a matter of fact, the saving is so great that most smelters use every effort to increase the proportion of metal-bearing material in their linings, and endeavor to counteract its inferior quality by increased care in the construction of the lining itself. It will, therefore, be scarcely worth while to devote space to the consideration of barren clay and quartz; for, if the student learns the method by which a lining made of inferior material may be made to endure reasonably well, he will have no difficulty in adapting his practice to the superior barren materials in case he is unfortunate enough to be obliged to make use of the latter.

Assuming the ordinary practice, the first question that will arise is the kind of quartzose ore most suitable for linings, and the size to which it must be crushed. Naturally we must use the type of ore which is the richest in free silica (quartz), and which is the freest from sulphides. This will be, ordinarily, the second-class ore from the mine, which would, unless thus employed, go to the concentrator; and the amount of such ore that we can employ for this purpose, although actually large, will usually be but a small proportion of its total.

The Montana smelters employed for a long time a highly silicious ore from the Snowstorm mine in Idaho. This ore is a quartzite, impregnated with copper carbonates and chalcocite, and averaging 85 per cent silica and 3.5 per cent copper. Being peculiarly adapted to the purpose, it soon commanded so high a price that resort was had to the ordinary second-class ore of the Butte copper mines and, while inferior in endurance to the Snowstorm ore, its use has now become universal. The following partial average analysis shows the more important constituents of the second-class ore used in converter linings at the Washoe smelter in the month of July, 1909:

SiO_2	65.32
FeO	8.12
S	7.77
Al_2O_3	9.97
CaO	0.28
	91.46

This ore also contains about 3½ per cent copper.

The size to which the quartzose portion of the lining material

should be crushed depends mainly upon the amount of fines which it produces during the crushing. There should be enough fines to pretty well fill the interstices between the larger fragments, and no more. This is usually accomplished by crushing to about the size of a hickory nut (0.016 m.). At the Boston and Montana smelter the crushed product is not even screened, the crushing-rolls having very strong springs. The substitution of the natural fines of the second-class ore for the crushed ore itself is not satisfactory, the former containing too much sulphides and too little quartz.

In place of the clay, highly aluminous ores may be used, such as the clay-gouge that so often borders the vein. In Montana, where the ores contain a decomposed feldspathic gangue, the fine slimes from the concentrator slum-ponds are found to be an excellent material. It consists of 20 to 25 per cent of actual kaolin, the balance being feldspar, quartz, sulphides, etc. A partial analysis shows approximately

SiO_2	60.	per	cent
Al_2O_3	18.	"	"
FeO	6.	"	"
S	6.	"	"
Cu	2.5	"	"
Ag	1.5 oz. —	"	"
	92.5	"	"

The incorporation of the quartzose and clayey constituents of the lining-mixture is executed in a Chili mill, or similar apparatus. Copper precipitate (cement copper) from the Butte mine waters is occasionally added to the lining-mixture. This substance cannot be added to the refining charges, as it is too arsenical, while, if worked up in the reverberatory or blast-furnaces, it makes too rich a matte.

During the first six months of 1909, the Washoe lining consisted of

special silicious ore	75.	per	cent
second-class ore	6.7	"	"
pond slum (from concentrator)	18.3	"	"
	100.0	"	"

Experiments having shown the advantage of substituting

second-class ordinary ore for the special silicious ore, the next month's mixture was

special silicious ore	31.0	per cent
second-class ore	53.5	" "
pond slum	15.5	" "
	100.0	" "

The elimination of the special silicious ore is still progressing.

The average analysis of the lining-mixture used at Anaconda for the first six months of 1909 is as follows:

SiO_2	75.70
FeO	4.29
S	3.56
Al_2O_3	7.13
CaO	0.32
Cu	4.08
Ag 6.97 oz. per ton	—
Au 0.072 oz. per ton	—
	95.08

The moisture in the prepared material will average about seven per cent.

At the Boston and Montana smelter, a lining consisting of 70 per cent second-class ore, 25 per cent pond slum, and 5 per cent copper precipitate was satisfactory. The following table gives an analysis, from the smelter laboratory, of a sample of the moist material, ready for use:

SiO_2	59.7
Al_2O_3	11.9
FeO	9.1
Cu	6.2
Moisture	6.9
	93.8

The lining material is dumped in layers by means of wheelbarrows, a mould for the cavity being set within the shell, and the layers of lining dumped into the annular space thus formed. The mould is usually of iron or steel plate, and is constructed in several pieces, for easy withdrawal. In certain of the converters the cavity is of such a form that some of the pieces constituting the mould must be wedge-shaped.

The tamping of the lining material into place is an important matter, as its duration depends largely upon this operation. In

all the larger plants in this country, the tamping is effected by machinery, a mechanism resembling an ordinary pneumatic rock-drill being employed.

A permanent lining of 9 inches of fire-brick is usually placed within the shell, and only needs renewal, or patching, when accidentally burned out. This is not an invariable practice, but is considered advantageous by the majority of technical men. The lining-mixture must be tamped in comparatively shallow layers in order to obtain the required firmness. This is especially essential if its quality is not of a high order.

At the Washoe plant the bottoms are tamped by means of a tamping-piston of 5 in. diameter and 20 in. stroke which is raised and lowered by an hydraulic piston, supported by a revolving jib-crane, while the drill for tamping the tops is suspended from a chain hoist. The vertical converters of the Boston and Montana smelter are placed upon an electrically operated turntable, and the tamping is effected by an especially-designed apparatus, with large cylinder, which is operated by the converter-air (16 to 18 lb. per sq. inch). The valve is operated by hand.

After the body of the vessel is lined, the tuyere openings are punched through, and the cap is put on. The latter is lined by hand, using the same mixture as for the body-lining, but made into balls. The space below the tuyeres, for the collection of the copper, is left about four inches deeper than the tuyere level. It wears deeper rapidly, but the metallic copper often submerges the tuyeres to a depth of several inches.

At the Boston and Montana smelter, ten men line three converters during an eight-hour shift, their work being divided as follows: 5 at the clay mill, 2 shoveling, 2 lining caps, 1 running tamping machine. The weight of one of these converters, lined, is 52 tons (47.16 m.t.). It costs here about $25 to re-line a vessel.

The following figures, kindly furnished me by the Anaconda Co., relating to its Washoe smelter, are of great interest, because they represent exact data covering long periods of time and unprecedented quantities of material:

tons matte blown per converter lining	50.479
tons matte blown per ton converter lining	3.563
tons copper produced per converter lining	18.063
tons copper produced per ton converter lining	1.275
tons copper produced per stall-day	29.621

Converters are dried before being moved, a fire of wood, coal, or coke being maintained in them for several hours, a perforated plug closing the air-channel through the trunnion, and being connected with a pipe which furnishes a light blast. One to two tons of coal or coke is required, and the time consumed in this operation varies from 8 to 24 hours.

H. L. Charles describes a method of drying linings[1] which he finds advantageous as regards both time and cost, as well as in lengthening the duty of the lining. This consists in pouring slag into the freshly-lined converter to form a shell of this substance which dries the lining and also protects and supports it until such aid is no longer necessary. The shell is provided with numerous vent holes for the escape of steam and water. The tuyeres of the newly-lined vessel are plugged on the inside with clay, and the converter is filled with slag — perferably from the process itself. After a period varying from one to three hours the slag is poured out, and the shell which has formed is allowed to cool gradually. In about eight hours it is ready for duty.

To allow for the thickness of the shell, the core used in lining the converter should be some two inches larger every way than the usual mould, and the bottom should be tamped about three inches deeper. This shell should be knocked off on the tuyere side. Charles states that, by the use of this device, he prolonged the life of his linings one to three charges.

At Anaconda, the top of the converter is tamped with the use of a mould, and is fastened to the body by means of bolts, slotted for wedges, the joint being made tight with clay. The old lining is often retained in part, it being necessary only to cut away enough to obtain a fresh surface to which the new material will adhere. The lining material of the Anaconda 8 by 12½ ft. barrel-converters weighs about 16 tons; the converter, with lining, 42 tons. A converter is lined there in about 1½ hours.

When the freshly-lined converter is to replace a worn one, it is taken by the electric traveling crane to the empty stand, where it is reversed in order to empty out ashes and debris, as soon as possible, and while still at a high temperature it receives its charge of matte from the smelting furnaces; the blast is turned on, and the vessel tilted back into its vertical position. At certain plants

[1] *Eng. and Mng. Journal*, June 1, 1907, p. 1046.

the vessel is not turned down to receive its charge of matte from the ladle, but is filled while still in a vertical position. In such cases, as at the Washoe smelter, the furnace-matte is poured from the ladle into a launder on the charging-floor above the converters. A section of the launder is constructed so that it can be swung over the mouth of the vessel, and a telescopic hood extends over the mouth of each converter to carry the fumes to the dust chambers.

As determined at Anaconda, the temperature of the blast-furnace matte, as it is tapped from the settler into the ladle, averages 1170 deg. C., that of the reverberatory matte, which flows some distance through an iron launder, 990 deg. C., while the temperature of the matte as it enters the converter is in the neighborhood of 900 deg. C., which is not far above the melting-point of the grade of matte under treatment.

Just before the charging, it is customary, at many works, to throw a few lumps of coal into the vessel to prevent explosions and shooting-out of the molten matte. These irregularities seldom occur if the initial temperature is sufficiently high, and result apparently (in an already-used converter) from a sudden liberation of sulphur dioxide due to the reaction of the fresh cuprous sulphide with copper oxides which have remained within the converter.

While the color of the flame and smoke issuing from the nose of the converter forms the indispensable guide for determining the condition of the charge, it is impossible to lay down any absolute laws regarding these phenomena, or even to give a brief general description that will apply to all cases. The reason is, of course, that small quantities of volatile impurities which may be contained in the especial matte under treatment exercise a strong influence upon the flame and smoke, and modify them to such a degree that a description which would be correct in every detail for one plant would scarcely correspond at all to the phenomena occurring at a neighboring smelter running on matte produced from different ores. I might, indeed, offer a description of the flame-phenomena exhibited by practically pure mattes; but this would, perhaps, be the most misleading of all, as such mattes seldom occur in practice. If one will contemplate, for a moment, the degree of elimination, by volatilization, of such common elements as lead, zinc, arsenic, antimony, etc., he will not be surprised at the almost total

obscuration of the modest flame-phenomena that characterizes the proper constituents of the matte — iron, copper, and sulphur. Each case is a law unto itself, to a considerable degree, and one must become familiar with local conditions before he can judge, with any accuracy, of the progress of the blow. After this local knowledge is once acquired, the condition of the flame enables one to judge, with much nicety, of the condition of the charge.[1]

With tolerably pure mattes a light greenish color is observable in the flame, although often streaked and obscured by dashes of yellow. As the ferrous sulphide diminishes, and the remaining matte begins to approach the condition of pure Cu_2S, azure-blue streaks are seen and, at the close of the period, the entire flame may have this beautiful blue color. All this, however, may be modified incredibly if the converter is too hot or too cold, or the layer of matte above the tuyeres is deeper than normal.

At the Washoe smelter, on a normal charge, the flame is bulky at the start, and passes through red, orange, and deep gold — followed by various shades of yellow and green — to a more or less azure blue; sometimes, however, to a pale-green with azure streaks. Toward the termination of this period of slag-formation, there is often a stage of great luminosity, associated with a green color and much white smoke. At this stage, some of the slag is usually poured off and, after a short further blowing, the smoke and luminosity disappear, the flame shows more blue, and the first period terminates with the complete pouring of the slag.

In the second, or copper-forming period, the main value attached to the color of the flame is that of an indicator of the temperature. The appearance of the flame during the initial stage of this period is obscured almost completely by the amount and quality of the charge of dope which has been added. It is also greatly influenced by the condition of the lining, the depth of matte above the tuyeres, and other causes.

Towards the completion of the period, however, these modifying influences disappear, and the normal flame should be thin, small, and of a pale golden color; a bright golden color indicates too high a temperature, while the reverse condition gives rise to orange or red tints.

[1] In referring to the flame phenomena at the Washoe converters I have the benefit of some notes taken by Mr. Hamilton Kilgour, who studied this point minutely.

As the end approaches, and the Cu_2S is nearly all decomposed, the flame becomes almost invisible, and the appearance of a spray of fine metallic sparks becomes the most noticeable feature. These are fine particles of metal ejected by the blast, and are an important indication of the condition of the bath. So long as these little globules consist of matte, they will adhere more or less completely to the wall of the flue into which the converter is blowing; so soon, however, as particles of metallic copper begin to be ejected, they will rebound and fall back. The condition of the copper adhering to the punch-rod will also indicate closely the purity of the charge within the vessel.

The degree of purity desired will determine how much longer the blowing shall continue, and the judgment of the attendant will determine closely when this degree is attained. The vessel is turned down, and the copper poured into moulds, or into the electric ladle to be conveyed, still in liquid form, to the refining furnace where it will undergo a partial refining to fit it for casting into smooth anode plates for electrolytic refining. At the Washoe plant the converter copper averages about 98.3 per cent copper, and this is raised to 99.3 per cent in the refining furnace. The copper from the latter is, however, purer than the above statement would indicate (for all practical purposes), because a portion of the 0.7 per cent which it lacks of being absolutely pure copper is made up of the oxygen belonging to the cuprous oxide that was formed during the refining process.

The various technical operations belonging to the blowing of a charge of matte are far too numerous and complicated to be studied in a work of this nature, nor can they ever be learned except at the converter itself. I know of no operation in the metallurgy of copper a knowledge of which is so difficult to acquire from books. I can merely indicate a few of the more important irregularities that may develop, and the customary measures in use for their neutralization.

The most common and serious error of the inexperienced is to fail to recognize the termination of the first period, and to go on blowing after all of the ferrous sulphide has been removed and slagged. This is likely to result in an explosion, which may scatter a large part of the liquid contents of the vessel over the entire shed with serious consequences. It is supposed to result from the formation of copper silicate, and the sudden reaction

of the same with the bath of cuprous sulphide, forming a great evolution of SO_2 gas at the point of contact between slag and matte.

Another manipulation, which is so common as to be almost a portion of the regular routine, is to cool the contents of the converter after the pouring-off of the slag at the termination of the first period. Under normal conditions the temperature in the converter is high at this stage and, if the second period were inaugurated with so great a heat, the cuprous oxide, which is constantly being formed (and as constantly being reduced to metallic copper by reacting with Cu_2S), would unite with the silica of the lining, forming a refractory cuprous silicate, which tends to obstruct the tuyeres and necessitates constant punching. In order to cool the bath of cuprous sulphide to the proper temperature it is customary to introduce two or three tons of "dope," by the aid of the traveling crane. "Dope" is the name applied to material fed into the converter in this manner for the purpose of qualifying, or "doctoring," the original charge. It consists of materials which are already so rich in copper that they will not "throw back" the white metal in the converter, and is usually made up of white metal, old linings, floor sweepings, etc.

If matters have gone badly during the first stage and the vessel is too cold, it may be raised to a suitable temperature by means of heavy sticks of wood, by the use of coal, or by pouring in a charge of the original furnace-matte on top of the white metal, and thus repeating the first stage. The combustion of the ferrous sulphide of the lower-grade matte furnishes a large excess of heat, and will restore the temperature without difficulty. This operation is called "doubling" a charge, and is of common occurrence, its use not being confined to the conditions just described. It is, for instance, employed when the charge of white metal is too small for economical work, as frequently happens in a freshly-lined converter.

Of late years the addition of quartzose ores to the charge during the converting operation has assumed importance at certain plants. Originally done to patch weak spots in a lining, it was extended with the deliberate purpose of protecting the existing lining by furnishing free silica for the fluxing of the ferrous oxide of the matte, and has now become an important means in working-up silicious ores at small expense.

While in use to a certain extent at many plants, I know of no place where this plan is employed so consistently and extensively as at the Boston and Montana smelter, at Great Falls, Montana. In order to avoid the annoyances arising from floating masses of half-fused material upon the surface of the bath, the added ore, at this smelter, is — so to speak — plastered onto the lining, whence it becomes gradually detached, and is eventually slagged by the ferrous oxide. This result is accomplished as follows: the empty vessel, still hot from the charge just poured, is tilted down, and about 5000 lb. (2268 kg.) of raw, silicious, second-class Butte sulphide ore is charged into the cap of the vessel from so-called "boats" handled by the electric crane.[1]

After the ore is charged, matte is poured over it, and allowed to stand two or three minutes, which cements the ore firmly in the cap. The vessel is then tilted back to its normal position and the blowing of matte proceeds as usual. The layer of ore is melted off little by little and, floating in small amounts upon the surface of the charge, is gradually slagged without interfering mechanically with the operation. The officials of the company state that it would be impossible for them to make anything like their present tonnage of copper per ton of lining were it not for this protective device, and it certainly seems to be an advance over our present unsatisfactory plan of using our furnace walls as flux for our ores.

The length of time required to convert the original matte into metallic copper depends upon the grade of the matte, the size of the charge, and the volume of the blast. How true this statement must be can be inferred from the fact that the examination of the converter gases, during normal running, shows that practically all of the oxygen of the blast has been utilized in its passage through the melted matte.

As pointed out by Sticht, there is a preferential oxidation of iron at the beginning of the blow, the escaping gases showing only one per cent, or thereabouts, of SO_2 for some little time. This is doubtless the excess iron which most mattes contain, and which results from the dissociation of FeS, at very high temperatures, by heat alone. While this formation of SO_2 is so peculiarly scanty

[1] If the bottom is too deep, the ore is charged accordingly, or may be placed at any accessible point where the lining needs building up.

it is significant that the gases contain only traces of oxygen; it is reasonable, therefore, to infer that, during the first few minutes of the blow, there is a great oxidation of metallic iron preceding the active combustion of the FeS. In certain reverberatory mattes, where the proportion of sulphur corresponds more nearly to the iron content, it is likely that the evolution of SO_2 would be more active from the very outset.

After the first 10 minutes the SO_2 of the gases runs up to about 10 per cent, and a trifle more oxygen is found, but not enough to show that there is any considerable escape of that gas in a free condition.

In the second period the SO_2 runs up to 18 per cent (by volume), with rarely over 0.5 per cent of oxygen; while at the finish the oxygen may run up to two or three per cent or more. It is quite evident, then, that in good practice almost the whole of the oxygen of the air is consumed in its passage through the matte. Indeed, it would be truly remarkable if this were not the case.

The importance of the free silica in the oxidizing reactions of the converter is well emphasized by Sticht in his presidential address already cited. He says: "What we call free silica is in reality simply the anhydride of an exceedingly strong acid, which becomes active only in an igneous condition, but then seeks its base the same as any other mineral acid. As far as its chemical relations to iron are concerned under the conditions existing in the converter, that element (iron) behaves exactly the same as it does at ordinary temperatures in the presence of ordinary free acids. Hydrogen sulphide will not precipitate iron in the presence of hydrochloric acid, for instance; and, similarly, iron will not form sulphides at high temperatures in the presence of free silica. Conformably, therefore, sulphur already combined with iron can be driven out and burned in an oxidizing atmosphere when uncombined silica is present, the iron and silica uniting.[1]

"One of the most striking features of the converting operation is that this union follows set lines, controlled by the heat generated within the converter and by the formation-heat of definite silicates. Only that particular silicate of iron will form which *can*

[1] Of course, the blowing may be overdone, as at the Garfield smelter where they bessemerized as rapidly with 11 lb. of air as with 16 lb., the column of matte being the same in both cases.

form at the temperature evolved, and none other. Against popular expectation, the formation of a silicate approaching a bisilicate is indicative of a lower temperature than the production of a singulosilicate. For the sake of the best utilization of the lining — or, in other words, its consumption to the smallest possible extent — a basic slag should be looked forward to.

"Fortunately, the thermal energy of copper-converting is vigorous enough to permit the generation of the singulosilicate of iron. Its formation-temperature has been synthetically determined as about 1270 deg. C., a figure which agrees with the direct measurement of the temperature in the converter, as also with the calculation of the same from analyses.

"The presence of scorifiable bases other than iron modifies the resultant heat and slag; but, in a general way, these remarks characterise the position. No variety of manœuvring will get more silica, or more iron, into the slag than the thermal conditions allow. The system of forces in the converter, being left entirely to itself, only creates that phase of results which is the easiest for itself."

Returning to the matter of time required for blowing matte to metallic copper, we see that it is modified by so many variable factors that it is impossible to state any general rule. The only way to obtain any really useful information on this point is to appeal to the results of experience, and I will add a few examples of the best practice in this country.

In the first place, when estimating the capacity of a given number of converter-stands (each one provided with extra vessels), it must be recollected that there is a good deal of unavoidable delay, so that it is not safe to count on over two-thirds of the theoretical capacity that the plant would have, could each stand be kept busy blowing during every moment of the 24 hours. Where there is abundant smelting capacity, so that there is little waiting for matte, the chief delays come from repairs which put a stand out of commission for a shift or more; there are apt to be serious delays in the over-worked lining and drying departments; the duties of the electric cranes are apt to be massed, so that they are unable to keep up with work at one period, and are comparatively idle at another; finally, frequent accidents, explosions, burst-outs of matte, etc., often force one of more

stands to remain idle for a long time. The following table calculated from one of the twenty-four hour reports at the Washoe smelter shows the total time lost, as well as the distribution of the same. During this period of 24 hours there were 11 stands in commission, although three of them stood idle for periods, respectively, of 14, 15, and 16 hours, owing to repairs and other causes.

If the 11 stands should blow continuously during the entire 24 hours, it would be equivalent to $11 \times 24 = 264$ hours blowing for one stand. Of this theoretical 264 hours, 34 per cent was lost, distributed as follows:

DISTRIBUTION OF THE TIME LOST IN BLOWING

charging the converters, etc.	7.94	per cent
waiting for matte	15.2	" "
changing converters (9 changes)	5.26	" "
stall down	71.6	" "
	100.0	" "

During this same period 74 charges of matte were blown, and 64 pourings of copper were made. There was also added to the converter-charges 34 boats of dope (handled by the electric crane), and 62 buckets of the same material (supplied by the small individual hoists adjoining the stands).[1]

Official figures from the smelter office, covering long periods, show that the average time required to blow one full converter-charge of their matte (40 to 45 per cent copper) is 157 minutes, of which 143½ minutes is consumed in the actual blowing. The time of blowing used per ton of matte converted is 19¼ minutes. The time required per ton converter-copper is 54 minutes. The air-pressure is about 16 lb. per square inch.

The charge for a freshly-lined Washoe converter is seven tons of matte. This is seldom increased to more than nine tons on a worn lining which might contain much more. It is preferred to use the doubling-system, by which a converter which already contains its own charge of white metal — resulting from the blow just completed — receives a second, or doubling, charge from another vesssel and blows the double charge to metallic copper.

The lining at Anaconda lasts for about six charges of matte, the first and last charge, however, being blown only to white metal;

[1] The "boat" averages about 1650 lb. net.

this is poured into some other vessel, where the lining is already sufficiently corroded to contain a double charge. The reason for adopting this practice with the initial charge is that it is too small in quantity to be blown to metallic copper economically, while the last charge is used as a "wash-out" charge before the converter is re-lined.

In a general way it may be said that the average time required at this plant to blow a charge of 40 to 45 per cent matte up to white metal is about one hour; to complete it to 98.3 per cent copper requires an additional 60 to 90 minutes.

The amount of dope (all kinds of rich material from the converters) charged back into the converters at Anaconda averages about 70 tons on a daily production of 300 tons metallic copper.

The situation at the Great Falls (Montana) smelter does not lend itself so well to statistical results, because it has been, for a long time, conducting extensive experiments as to the best size and form of converter, and now has in use no less than four types.

The older small vessels, 7 ft. diameter.
No. 2 converters, 8 and 10 ft. diameter, the short diameter being on the trunnion section.
No. 3 converters, 10 and 12 ft. diameter, the long diameter being on the trunnion section.
No. 4 converters, 12 ft. diameter, circular.

Including all these types of converters, they average a production of about 29 tons (26.3 m. t.) of copper per stall-day, which is about the same as at Anaconda.

The bessemerizing at Cananea, Mexico, is interesting on account of the low grade of the matte converted. De Kalb gives some interesting official [1] statements, from which I abstract the following information: there are five converter stands, the vessels measuring 96 × 135 in. (2.44 × 3.43 m.), with 14 cast-iron tuyeres, $1\frac{1}{2}$ inches (0.038 m.) in diameter. The blast-pressure is 12 lb. at the power house. The charge varies from four to seven tons of 30 to 32 per cent matte, which is blown to 60 per cent in from 40 to 60 minutes. Sufficient raw matte is then poured into the converter to make a proper finishing charge, which is poured as 99.1 blister. Each stand averages about 43.5 tons matte per day, and a lining suffices for 24 tons raw matte, or 8.9

[1] "Cananea Furnace Practice," by Courtenay De Kalb, *Mining and Scientific Press*, July 2, 1910.

tons blister. The consumption of lining is about one-third of a ton per ton of matte. A very compact lining is made from an ore produced at the company's mine, having the following composition:

moisture	5.6	per	cent
SiO_2	56.00	"	"
Al_2O_3	14.20	"	"
FeO	5.40	"	"
CaO	1.70	"	"
S	5.50	"	"
Cu	2.49	"	"
	90.89	"	"

I have already mentioned the extremely basic character of the Cananea slag produced during 1909, due to the solidity of the converter linings, and the low grade of the matte converted. This has gone still further until, in April 1910, the converter slag had the following unusual average composition:

SiO_2	21.5	per	cent
Al_2O_3	7.4	"	"
FeO	60.9	"	"
CaO	1.2	"	"
S	1.1	"	"
Cu	2.5	"	"
	94.6	"	"

It must be recollected that this slag results from blowing the low-grade matte up only to 60 per cent copper, instead of to white metal.

Ten to 13 shells are lined per day, one converter-lining requiring 2.93 bbl. of Oklahoma crude oil to dry it in 10 hours.

The matte ladles, of cast steel, have a holding capacity of 68 cu. ft., and last for about three years.

The total air requirements, in terms of horse-power per ton of raw matte (30 to 32 per cent copper) produced and charged are, for the entire furnace plant: blast-furnaces, 0.0651 h.-p.; converter-blowing, 0.0728; converter-rotating, 0.00098; crane-work, 0.00207. The converters take 5230 cu. ft. air per minute per converter, or 220,000 cu. ft. per ton of bullion produced.

The converter-house is served by one 40-ton and one 50-ton crane, with 50-ft. span, requiring 13.6 h.p.

The converter gases are conducted through a four-foot flue to a stack five feet in diameter.

The amount of slag produced in converting matte depends mainly upon the grade of the matte itself. Every pound of iron which the matte contains will produce 1.82 lb. slag (assuming that there is ample heat in the converter to ensure the formation of the normal unisilicate of iron), which weight will be still further increased by the earthy constituents of the lining. This refers, of course, solely to the slag produced during the first period of the blow, and which is poured off before the stage of copper-blowing is begun. The slag from the latter stage need not be considered at all, as it is worked up in the converter itself usually by allowing it to chill and form pellets or granules, which float upon the metallic copper and are kept back with a pole or rabble while the copper is being poured. These granules are extremely rich in oxidized copper, and usually contain a nucleus of quartz from the converter-lining. They are either allowed to remain in the converter, to be fluxed into first-period slag during the succeeding blow, or are dumped upon the floor and added to another matte charge. Thus this second-period slag is worked up in the converter process itself, and does not concern us.

Comparatively little attention is paid to the composition and copper content of the main slag-product from the converters, because it always contains too much copper to be thrown away and, as it must be re-treated, a little copper more or less in it makes no serious difference.[1]

At Anaconda, where the matte averages about 45 per cent copper, its assay in copper for one period of six months was 2.39 per cent copper. This was almost entirely present in the shape of prills of white metal in mechanical suspension, showing how extraordinarily the copper is protected from slagging while sulphides are present.

This slag here, and at most plants, forms a welcome addition to the blast-furnace charge, both from its ferruginous character and its influence in keeping open the ore-column. It is poured from the converter into cast-steel ladles, unlined and having a capacity of five tons, which stand on the floor. When one of the electric cranes is at liberty, it places the ladle in a hydraulic cradle

[1] A 45 per cent matte usually produces about 65 per cent of its weight of slag.

which pours it into traveling moulds of a casting machine. The conveyor dumps the slag-cakes, sprayed with water on the way, into steel bins, where it is loaded into the cars which convey it to the blast-furnace storage bins. This arrangement is typical for those plants at which a ferruginous slag in lump form is welcome in the ore-smelting portion of the process. In such instances it is estimated that sufficient benefit is derived from the addition of this slag to cover the cost of its treatment.

It will be noted that the converter slag, while highly ferruginous, is only slightly basic.

In general, it contains in the neighborhood of 36 per cent silica and about 2½ per cent copper, when poured with care.

The following analyses give a general idea of its composition, the Parrot slag, however, being abnormally low in copper, and probably having settled somewhat before the sample was taken:

CONVERTER SLAG FROM FIRST PERIOD

	Parrot 50% matte	Anaconda 45% matte	Cananea av. (1909) 30% matte
SiO_2	36.80	35.70	26.8
FeO	50.40	55.83	58.3
MnO	—	0.22	—
Al_2O_3	6.80	1.76	6.2
ZnO	4.43	0.86	—
S	0.47	1.03	0.8
Cu	1.0	2.14	2.44
	99.80	97.54	94.54

The basicity of the Cananea slag is due to unusually solid linings.

When this slag is not desired, in lump form, at the blast-furnaces, the main object in its further treatment will be to settle out of it the prills of white metal that it contains, and to separate as well such small proportion of the copper as may be actually combined with silica or, at least, dissolved in the slag.[1]

The former of these purposes is attained quite thoroughly — the latter, tolerably — by pouring the slag into the hearth of a

[1] I cannot touch upon the solubility of metallic sulphides in slag, in this work. It is a subject worthy of closer attention than it has yet received.

reverberatory smelting furnace, or into one of the large settlers, or forehearths belonging to the blast-furnace. In a few instances it has been poured into the feeding-door of the blast-furnace with satisfactory results.

At the new smelter at Tooele, Utah, which has all the benefits of modern experience as interpreted by some of the best metallurgical skill of this country, the converter slag is poured into the large reverberatory smelters.

The granulated slag, from the period of copper-blowing, is so variable in both its physical and chemical composition, that attempted average analyses are misleading rather than instructive. This is of little moment, as it is worked up continuously in the converter operation itself.

The removal and replacing of the vessels may be effected, in small works, by means of a car with low platform which is run under the vessel. The platform of the car is then raised by an hydraulic piston, and the vessel thus lifted from the stand and carried away to be lined.

The best and almost universal agent, however, for executing this manœuvre, as well as all the rest of the heavy work about the converters, is the electric crane. This is too familiar an apparatus in industrial establishments to demand description. I may say, however, in a general way that a crane should be selected of a capacity exceeding by at least 50 per cent any duty which it is to be called on to perform. Thus, if the heaviest, freshly-lined converter is to weigh 40 tons, the crane should have a safe lifting capacity of 60 tons. This margin of safety is essential to avoid repairs and break-downs.

A large plant will also have smaller cranes in the lining-shed to facilitate the heavy handling of which there is so much.

At the Washoe smelter, with 11 converter stands, there are two 60-ton electric cranes in the main building, as well as two 15-ton electric cranes for the casting department.

The plan and general equipment of a modern converter plant is so complicated and extensive a subject that it is impossible to attempt its consideration in a work of the scope of this volume. A study of existing plants should always precede new construction. Aside from the large collection of detailed plans carried by the manufacturers of smelting machinery for the benefit of prospective customers, there is a considerable amount of valuable literature

upon the subject scattered through the principal technical journals and the transactions of societies.[1]

Although the bessemerizing of nickel mattes cannot be considered in this work, the behavior of this metal, when present together with copper, presents interesting features that may receive a brief mention.

The Canadian Copper Company, of Sudbury, Ontario, having been engaged for several years in bessemerizing mattes containing some 20 per cent each of copper and nickel, has presented an unusual opportunity for the study of this subject, and David H. Browne, metallurgist of the company, publishes some interesting material concerning this little understood matter.[2] Among the conclusions which he draws from a five years' study of the local phenomena of bessemerizing are:

1. Nickel is not an element replacing copper in matte.
2. Nickel-copper alloys act in the matte-blow as one metal.
3. Nickel-copper alloys follow, in the matte-blow, exactly the same laws as govern the behavior of copper alone.

Experiments at this plant show that, in bessemerizing nickel-copper matte, nickel does not follow the iron into the slag; that only in the last stages of the blow does it oxidize; that when it does pass into the slag, it carries more than its own weight of copper with it; and that it is impossible to blow all of the nickel out of the matte. The elimination of the sulphur and iron is, therefore, a direct function of copper-nickel in such mattes, and not of either metal separately.

The Mountain Copper Company, in using small converters (68 × 96 in.) at Martinez, California, found it advantageous to employ crude oil instead of coal or wood to develop heat in a cold charge, or to prevent the shooting or foaming of a fresh charge.[3]

The oil is conducted to the rear of each converter stand through a one-inch pipe, under a pressure of 20 lb., preheated by laying the pipe alongside the steam boilers. The flow of oil is controlled by a valve, and it is introduced through the wind-box into one of

[1] In the *Eng. and Mng. Journal* of March 17 and 31, 1904, C. H. Glasser gives much useful information as to converters, cranes, accessories, etc.

[2] *Bulletin of the Am. Inst. of Mng. Engrs.*, April, 1910.

[3] "Use of Crude Oil in Converters," by W. B. Rountree, *Eng. and Mng. Journal*, Oct. 5, 1907.

the tuyere holes. The pipe is reduced to one-fourth inch, and no further burner is used, the oil not being atomized. The same device is found economical for drying newly-lined converters.

A procedure of both scientific and technical interest was carried out a few years ago in the converter plant of the British Columbia Copper Co., in the Boundary District, B.C.[1]

In order to carry out a contract with another smelter, this plant was forced to do a considerable amount of converting at a period when it was rebuilding its entire blast-furnace system, and had, in consequence, no furnace available in which to melt the accumulated matte for the converters, which were of the barrel type, 84 × 126 in. (2.13 × 3.2 m.).

It was decided to melt the matte in the converter itself, and the following method of procedure was instituted: a wood fire was lighted in the cold vessel, about 1500 lb. (680 kg.) coke dumped upon it, and a light blast admitted. When the coke became thoroughly red-hot all through, about three tons of matte was placed on top of it, and full pressure of air turned on (10 lb. pressure). The matte melted quickly, and more was added at once to increase the volume of the charge. This operation was repeated with occasional removal of the slag, until the charge was large enough to admit of blowing to a finish. From the time of commencing to fire the cold shell until the first blister copper was poured, about 3½ hours elapsed under normal conditions. About 15 tons of 40 to 50 per cent matte per stand was thus converted in one shift. After considerable experience it was found that 45 per cent matte could be converted easily in this manner, and finished hot. Matte up to 55 per cent in copper could also be handled by using care, although the higher grades required the occasional addition of coal to keep up the temperature.

The amount and pressure of blast for the converter process varies greatly at different plants. As already intimated, the original purpose of the barrel type of converter was to provide a simple means for maintaining a uniform depth of matte above the tuyeres, so long as possible, and thus to permit the use of a lighter blast. The partisans of the barrel converter now frequently use

[1] "Copper Converter Melting its Own Matte," by E. Jacobs, *Eng. and Mng. Journal*, Sept. 8, 1906. I make free use of this description.

16 and 18 lb. blast per sq. in., while those who favor the vertical converter confine their tuyeres to only a portion of the circumference (usually about one-half).

At the Washoe works, using the large barrel converters, with 16 lb. pressure, the consumption of air averages 188,500 cu. ft. per ton of blister copper. At the Mount Lyell smelter, using the small, Stalmann vessels, with a pressure of eight pounds per square inch, Sticht reports the average consumption of air over a period of two years, as 165,423 cu. ft. per ton blister. It must be recollected that the former plant is several thousand feet higher above the sea than the latter. At Cananea De Kalb reports the air-consumption as 220,000 cu. ft. per ton blister.

The bessemerizing being a midway process, with products going in both directions, and often with its flue-dust mixed with that of other furnaces, it is difficult to arrive at an accurate determination of losses. We may, however, at least say that the loss in copper is exceedingly small, that in gold unrecognizable in most instances,[1] and that in silver less than has always been feared in former years.

It is doubtful if records at a smelter have ever been kept more carefully than at Mount Lyell, and the comparative absence of custom ores and general simplicity of the mixture makes them more than ordinarily safe. Sticht reports as follows regarding the recoveries in the bessemer operation at that plant:[2] the

GRADE OF MATTE AND BLISTER COPPER

Since the beginning, in 1897	copper p. c.	silver oz. per ton	gold oz. p. ton
average grade of blister	98.82	85.68	2.955
average grade of matte	44.21	36.31	1.236

recoveries in the converter, *per se*, crediting the full content of the middle products, such as converter slags, linings, scrap, etc., are: copper, 98.83 per cent; silver, 99.32 per cent; gold, a plus. The recoveries, after debiting the losses incurred in the blast-furnace

[1] The extremely small actual loss in gold is often converted into an apparent gain, due to unassayable amounts of that metal in some of the materials treated.

[2] "Mining and Smelting at Mount Lyell, Tasmania," by Robert Sticht. *The Mineral Industry*, Vol. XVI.

treatment of the slag and linings, are copper, 97.98 per cent; silver, 98.82 per cent; gold, a plus.

I have notes of many cases where there has been a large loss of copper, reaching as high as six per cent, during the converting operation. These examples, however, had a totally insufficient opportunity for the settling of the mechanically suspended metallic particles that are carried out of the converter by the blast.

This may amount to a more serious loss than one would suppose from the nature of the material and from the fact that so great a proportion of the particles carried out by the blast fall back upon the converter floor, adhere to the hood or cap, or are otherwise deposited in comparative safety.

At Anaconda, where 11 converter stands blow daily about 670 tons of 45 per cent matte to 98.3 per cent blister copper, the weight of dust settling in the converter flues amounts to about 140 tons per month, containing 71 per cent copper and 12.5 per cent sulphur. This material goes to the ore-smelting reverberatory furnaces.

From an analysis of a considerable number of cases, as well as from personal experience in former years, I am inclined to repeat the statement as to losses (with average mattes) made in my old "Modern Copper Smelting" (seventh and later editions): namely, that with suitable dust chambers, and few volatile metals in the matte, the loss in copper will not exceed one to one and one-half per cent, while that of silver will run about one per cent higher.

Hixon, referring to the Anaconda plant which preceded the present one, placed the converting losses at one per cent of the copper and two per cent of the silver.

I know of no authentic case where the operation is on a sufficient scale to permit reliable deductions that controvert the above statements, and I think that American metallurgists are correct in regarding the bessemerizing of reasonably pure argentiferous and auriferous copper mattes as a process that yields peculiarly satisfactory results in the recovery of these metals. Taken in conjunction with the electrolytic refining of the converter copper, it seems to be by far the most satisfactory method extant for treating the type of mattes just mentioned.

With mattes containing much lead or zinc there is a greater opportunity for a divergence of opinion in regard to the loss of silver in the operation. Even this doubt, however, is not suffi-

cient to deter any one from using the process for mattes high in volatile metals, as the addition of a suitable bag-house will apparently reduce the silver losses to a very minute figure.

In about 1894 I received a considerable amount of data from Pueblo, Aguas Calientes, Swansea, Freiberg, and one or two other points interested in the bessemerizing of impure argentiferous mattes. Most of these experiments were made in apparatus much smaller and much less complete than would be used in actual work, and it is possible that the respective experimenters, including myself, were too ready to draw inferences from these results. At any rate the loss in volatilized silver was high in every case, and beyond all reason in one of them (conducted, as it happens, by the most experienced metallurgist of all of those to whom I refer).

James Douglas, in his paper before the Institution of Mining and Metallurgy, in London, 1899, says: "The loss of silver is admittedly large where zinc and lead in any notable quantity are present. At the Aguas Calientes works in Mexico, where copper is used as the absorbent of gold and silver, and dry silver and gold ores are smelted with copper ore and matte into matte to be subsequently converted, no ore with an excess of five per cent of lead and zinc is admitted to the copper furnaces. Experience at these works, where the contents of purchased ore has to be accounted for in the finished products, has fixed these percentages of the two dangerous associates as the maximum that can be safely allowed."

The introduction of the bag-house has changed all this; but later experience seems to indicate that, even without the bag-house, the bessemerizing operation as practised at present may not necessarily cause such serious silver losses from leady mattes as was formerly believed.

Moore gives some interesting results of experiments at Aguas Calientes,[1] as well as at Omaha, and I repeat here some of his conclusions:

At Aguas Calientes an extensive flue system and bag-house were installed in order to permit the introduction of more lead and zinc ores into the copper charge. When making matte with 200 to 300 oz. silver per ton, and carrying five to six per cent lead, the

[1] "Basic Lined Converters for Leady Copper Mattes," by Redick R. Moore, *Eng. and Mng. Journal*, Aug. 6, 1910, p. 263.

dust from the bag-house contained 50 to 62 per cent lead, and 10 to 20 oz. silver per ton. As the flue-dust collected in the bag-house contained less than one-third of the total lead in the matte, the silver loss would not have been over one-half of one per cent even had it been permitted to escape from the stack. The bag-house dust showed from a trace of copper up to one-tenth of one per cent.[1]

Moore goes on to state that the results obtained at Aguas Calientes led the company to install converters for leady mattes at Omaha, in 1905, also with bag-house. In converting these mattes, containing 150 to 200 oz. silver per ton, the bag-house flue-dust contained over 50 per cent lead, but not to exceed six ounces silver per ton. Moore finds from experience at these works that, aside from the losses in the slag-treatment, there should be practically no shortage in metals in converting leady mattes, with bag-house attachment, excepting the slight loss in cleanings and sweepings, which should be recovered for the most part.[2]

In the light of the experience gained at Aguas Calientes and Omaha I think it is safe to say that our former views as to the loss of silver in converting mattes containing considerable lead and zinc must be modified.

It is probable that, with suitable settling flues, this loss will be small, and that it may be practically eliminated by passing the converter gases through bags. The arrangement and care of such a bag-house will, however, require special attention, owing to the nature of these gases.

Almost all of the modern works are now arranged so that the matte is never allowed to solidify between the ore furnace and the converter. When feasible, it is convenient and economical to tap it direct from the furnace into the converter. This is, however, seldom attainable, and it is usually tapped into a lined sheet-steel ladle, mounted on trucks, or handled by the electric traveling crane. The deep hearth of the reverberatory and the large settler of the blast-furnace are the reservoirs which store or deliver the melted matte according to the demands of the converter plant.

[1] One ounce per ton = 0.003429 per cent.

[2] It need hardly be said that special precautions must be observed in order to prevent the destruction of the bags by the converter fumes. This matter is discussed in another chapter.

At a few works, however, the matte from the ore furnaces is re-melted in a small separate furnace, like a foundry cupola, and while the simple smelting furnace itself need not occupy our space, a word as to the cost of this re-melting operation is required.

Mount Lyell happens to be one of the great smelters where this re-melting is practised, and, as the Mount Lyell records are kept with extreme minuteness and represent large quantities of material under peculiarly regular conditions, its costs are most interesting; they are particularly suitable to Americans because both labor and supplies, in this remote portion of Tasmania, correspond more closely to our own standards than do most foreign conditions.

Sticht gives the re-melting costs of matte at Mt. Lyell as approximately 0.2235 cts. per pound of fine copper produced, or $2.01 per short ton of the 45 per cent matte. This includes its full proportion of general expenses.

It was about $2.08 per ton at the Parrot smelter, at Butte, in 1894.

Hixon gives former cost of re-melting matte containing 55 per cent copper in Montana, with coke at $12 per ton, as $2.20 per ton of 55 per cent matte, or 0.2 cts. per pound copper in the matte.

It is, therefore, safe to assume that with present prices and a plant of fair size, matte may be re-melted for the converters at accessible points, for $2.00 per ton (0.907 m. t.), or less than one-quarter cent per lb. copper, in a 45 per cent matte.

The costs of the complete converting operation cannot be given satisfactorily in a brief statement, owing to the extremely variable conditions which obtain at different plants. The most important item of all — the cost of power for the blast — influences the total expense so greatly that no fair comparison can be made without reducing all of the plants to a common standard, and this is also true, in a lesser degree, for the other items.

Hixon gives the former cost of converting a 55 per cent matte at Anaconda at about $4.95 per ton (0.907 m. t.), or 0.45 cts. per pound copper.

I estimated the cost of converting a ton of 50 per cent matte at the Parrot smelter in Butte, in 1894 (omitting re-melting) as $5.92, or 0.592 cts. per pound copper.

Sticht's careful analysis of the costs of converting at Mount

Lyell, modified slightly by my own attempt to deduct the expense of re-melting, is too valuable to omit:

ANALYSIS OF COSTS OF CONVERTING MATTE AT MOUNT LYELL
(including re-smelting of the matte)

	Cents per pound fine copper in 45 p. c. matte	Dollars per ton matte (2000 lb.)
re-melter labor	0.0454	0.40,86
re-melter coke	0.1281	1.15,29
converting labor	0.1427	1.28,43
lining labor	0.0792	0.71,28
lining materials (clay, quartz, etc.)	0.0186	0.16,74
stores and supplies	0.0286	0.25,74
flue-dust (handling, clinkering, etc.)	0.0160	0.14,40
power (fuel, labor, and stores)	0.1942	1.74,78
dressing and sampling blister	0.0474	0.42,66
loading blister into trucks	0.0024	0.02,16
general expense (administration, laboratory, maint., etc.)	0.0490	0.44,10
total	0.7516	6.76,44
deduct re-melting costs (in part estimated by E. D. P.).	0.2235	2.01,15
cost of converting without re-melting	0.5281	4.75,29

At one of the great Montana copper-smelters, the annual distribution of costs for the year 1905 allowed 0.39753 cts. (practically four-tenths of a cent) per pound copper for the cost of converting a 45 per cent matte. This is $3.58 per ton of matte.

A modified converter, called the "Selecteur," invented by M. Paul David, is intended for making metallic "bottoms" during the operation of bessemerizing, the object being the extraction of the gold and the removal of objectionable impurities from the matte, as in the Welsh blister process when bottoms are made.

The device consists, in the main, of a pocket on one side of the shell, into which the metallic copper drains when the converter is turned into a suitable position after the formation of metallic copper has continued for a proper time. The pocket is so shaped as to retain the metallic bottoms, which can be tapped from it after the converter has been returned to its vertical position to complete the blow.

It is unfortunate that we have started by applying a misnomer to a piece of apparatus which is attracting considerable attention at present, and which may play an important part in the metal-

lurgy of copper. The so-called basic-lined converter[1] is not lined with basic material because *basicity* exerts any useful influence in the bessemerizing operation. We simply desire to avoid a silicious lining so that it may not be fluxed away and destroyed by the ferrous oxide of the matte and, thus far, it happens that a basic material has been found to be the cheapest and most suitable to take its place. It might just as well be a neutral lining, or an acid lining, if we could prevent its being eaten away by the ferrous oxide.

Magnesite brick presented itself to the minds of the earliest experimenters (Keller, in 1890, and others later) as the most suitable non-silicious and refractory material, but failed completely in the small converters then in use. The three chief difficulties experienced in these early trials were: the conductivity of the magnesite, which cooled the interior of the vessel; spalling and disintegration of the brick from changes in temperature; impossibility of fluxing the FeO at all satisfactorily with added silica.

The first and second of these difficulties have been overcome by improved construction; the third is obviated by conserving the heat more completely than is possible in the ordinary converter and, especially, by working on a large scale and with a great volume of molten material. As I have pointed out more than once in this volume, a metallurgical operation which is a complete failure on a small scale may present scarcely any difficulty if repeated with a sufficient weight of molten material to smooth out irregularities and ensure uniformity.

Omitting, for the moment, Knudsen's magnesite-lined converter, which he uses for the two-fold purpose of ore-smelting and converting, we may consider the Baggaley process, which operated on very low-grade matte and silicious ore in Butte, 1905–1906, using magnesite brick as a lining.

[1] In the following attempt to give some idea of the present status of the basic-lined converter I disclaim all originality. My own personal experience in the matter is confined to a few unsuccessful experiments in the early days of bessemerizing copper. My material is gathered largely from personal communications which I am not at liberty to acknowledge, from a description of the Pierce & Smith converter, by Richard H. Vail, in the *Engineering and Mining Journal* of March 12, 1910, and an article entitled "Copper Converters with Basic Lining," by Redick R. Moore, *Eng. and Mng. Journal*, June 25, 1910. I make especially free use of the latter paper without constant acknowledgment.

Baggaley's converter was made up of steel rings 11 in. thick and 24 in.[1] wide. The outside length is 13 ft.; outside diameter, 8 ft. It is lined with nine inches of magnesite brick, the internal length being 9.5 ft., and the diameter, 4.75 ft.

The general plan of working is about as follows:[2] one thousand pounds of silicious ore being placed in the converter, about five tons of 28 per cent copper matte — obtained by the raw smelting in a blast-furnace of low-grade sulphide ores — is poured in and the charge blown in the usual manner until most of the iron has been slagged, silicious ore being added as required. The slag being poured off, more low-grade matte is added and the operation repeated until a sufficient body of white metal has accumulated to warrant blowing for blister copper.

Where there is a good margin of profit on silicious ores, Baggaley prefers, for a basis, matte of as low a grade as 10 per cent copper.

Heywood reports that from Oct. 7, 1905, to Jan. 31, 1906, over 480 tons blister had been made, and estimates that the converter could produce 500 tons blister per month if it could be supplied with matte.

He states also that the same magnesite lining was in use that had been put in at the commencement of the work, the only repairs required being around the tuyeres, where the brick had been destroyed by punching.

Baggaley has taken out a large number of patents covering mechanical details; this is not the place to discuss their value, but I conceive that he is entitled to decided credit for showing conclusively that the silica, needed to flux the ferrous oxide of mattes, can be furnished successfully by the addition of quartz ores, and without relying in the least upon an acid lining.

The question of the economy of this method of procedure is an entirely different matter. In my opinion the blast-furnace is a much more efficient and economical apparatus for smelting ores than is any form of converter, and I can see no advantage in trying to curtail the duty of the cheap continuous apparatus in order to throw more work onto the expensive periodic portion of the plant.

[1] One foot = 0.305 m. One pound = 0.4536 kg. One ton = 0.907 m.t.

[2] See letter by W. A. Heywood, technical manager of the plant, *Mining and Scientific Press*, April 28, 1906.

After a few months of experimental running, the Baggaley plant closed its operations, having demonstrated the suitability and stability of a lining for converters of magnesite brick, and the feasibility of supplying the needed silica by adding silicious ore to the bath of matte.

The legitimate use of the converter begins, of course, at that point where it can oxidize the sulphides of the material under treatment more cheaply and advantageously than the smelting furnace can do it. The exact percentage of copper that marks this division line must necessarily vary with local prices and conditions, as well as with the experience and opinions of different observers. So far as I can judge at present, I should place it between 30 and 50 per cent, under anything like ordinary conditions. Thirty per cent at the one extreme, because the converter is not an advantageous type of furnace in cases where there is too much valueless product in proportion to the metal present; fifty per cent at the other extreme, because this is about the highest limit to which one can go in ore smelting without re-treatment of the slag.

Coming then to the development of the legitimate copper converter, it seems to me that the efforts of Smith and Pierce, at the plant of the Baltimore Copper Smelting and Rolling Company, represent the first successful use, on a continuous commercial scale, of the basic-lined converter for the bessemerizing of ordinary copper mattes. That this is not too sanguine a view to take is evinced by the subsequent instalment of a similar converter at the Perth Amboy, New Jersey, smelter, and the erection of five such converters — after long and careful experimenting — at the Garfield smelter in Utah.

Three of the chief difficulties experienced in the operation of a converter lined with magnesite brick are: the difficulty of maintaining a sufficient temperature, owing to the conductivity for heat of this material; the mechanical destruction of the lining from expansion and contraction, and other strains; the rapid mechanical destruction of the lining about the tuyeres, due to the excessive punching demanded.

A brief study of the apparatus and its action will elucidate these points, and will also show how they have been overcome far enough to allow the process to proceed with sufficient economy to

satisfy competent metallurgists that it is — at least, under certain local conditions — a distinct improvement upon the ordinary method.

The accompanying illustrations show the type of basic-lined converter now in use at the Perth Amboy and Garfield smelters, and much of the following description is from the papers of Vail and Moore, already cited.

The shell is 23 ft. long and 10 ft. in diameter, the oval opening, after the lining is in place, being 6.25 ft. high, 7.25 ft. wide, and 19.75 ft. long. The shell is constructed of $\frac{3}{4}$-in. steel plates, excepting the ends, which have a thickness of only one-quarter inch. In order to permit the rising of the lining from expansion, the shell is open at the top, being secured by tie-rods.

It has thirty-two $1\frac{1}{4}$-in. cast-iron tuyeres, each with its separate Dyblie valve, and flexible hose connection to the air-main.

The three heavy flange-rings that surround it rest on trunnion wheels having movable bases, so that they may be adjusted to equalize the 100-ton weight which they have to carry when the vessel is well filled with matte.

Two openings, designed for charging matte and ores, are shown in the accompanying side sectional elevation, but have now been abandoned, everything being charged through the passage for the gases. This main circular throat-opening is at one end, and is about 40 inches in diameter. It is of sheet iron or steel, and lined with clay, for converting ordinary mattes.

The single opening for the removal of the molten products is in the side of the cylinder opposite the tuyeres. It is closed with clay and magnesite brick, and is only removed in its entirety when pouring copper, a smaller opening in this breast being sufficient for the exit of the slag.

An oil-burner is provided at one end, to restore the temperature if the vessel is running cold, or to dry the interior after repairs.

The lining is of 9-inch magnesite brick, except at the tuyeres, where 18-inch brick are employed. The bottom-filling is of fire-brick, and is about 13 in. thick in the middle, tapering to four inches at the sides. The magnesite brick are laid in dry magnesite powder, excepting at the region of the tuyeres, where a mixture of magnesia and linseed oil is used. Further experience shows that this does not form a permanent cement.

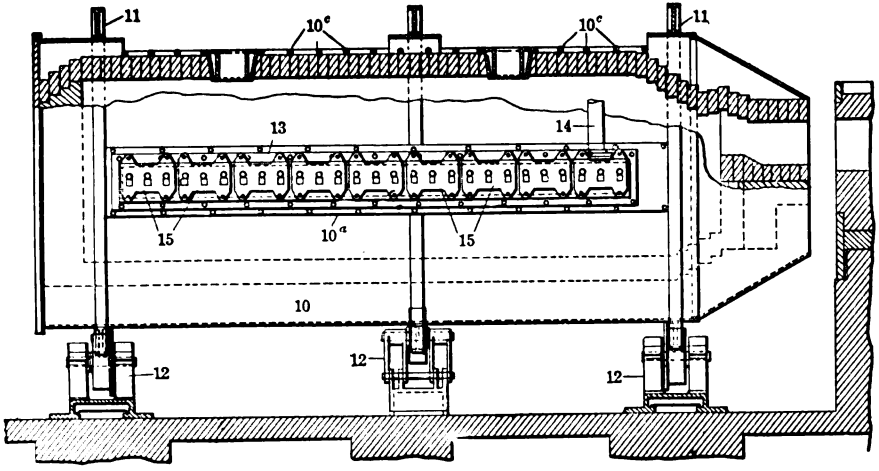


FIG. 95. — Basic-lined converter — side sectional elevation

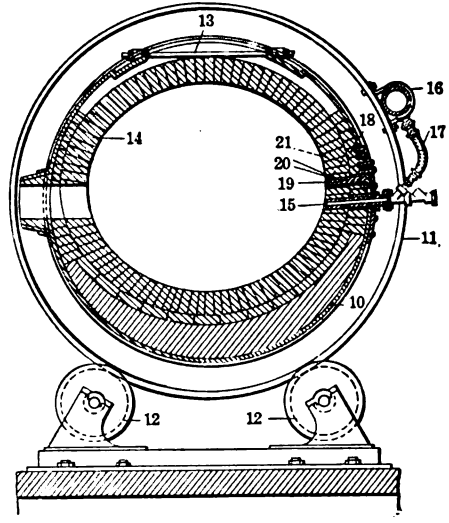


FIG. 96. — Basic-lined converter — individual tuyere pipes

The tuyeres are arranged in a manner supposed to ensure a certain degree of flexibility, in order that they might not suffer from movements of the lining. This is not believed to be essential, nor was trouble experienced from this cause in the Baggaley converter.

The operation in this converter differs but slightly from the ordinary process as carried out in acid-lined vessels. In starting, the interior is heated by the oil-burner, or other appropriate means, for some 24 hours, when an initial charge of four or five ladles (40 to 50 tons) of matte is poured through the throat, 6000 lb. of dried silicious ore is added, a light blast is turned on, and the converter tilted into position until the tuyeres come well below the matte. After the ore has spread itself well over the surface,

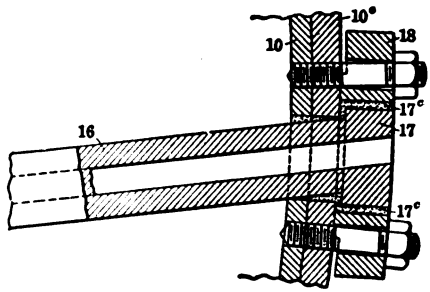


FIG. 97. — Basic-lined converter — the cast-iron tuyere

the blast is increased to 6000 or 8000 cu. ft. air per minute, engine displacement. With 24 inches of matte above the tuyeres, this gives a pressure of 10 to 12 lb. per square inch. In 30 to 45 minutes the silica will have been fluxed by the ferrous oxide, and the slag is poured off in the usual manner. Another ladle of matte, with corresponding silicious ore, is added and the blowing continued. This is kept up, with decreasing weights of silicious ore, until a large amount of concentrated matte has accumulated, when it is blown to blister in the ordinary manner. Tolerably clean copper mattes can be blown from white metal to blister without interruption.

The slag from this process is generally somewhat lower in silica than the slag made in acid-lined converters. Free silica ores are scarce at Garfield, and the addition of silicious ore is kept as low as possible. If the silica is reduced too far, the higher

oxides of iron are produced, and form accretions which are difficult to remove. The lowest practicable limit in silica for this work appears to be about 25 per cent, the converters at Garfield usually making a slag with from 27 to 28 per cent silica. Owing to the high atomic weight of lead, the silica in slags resulting from

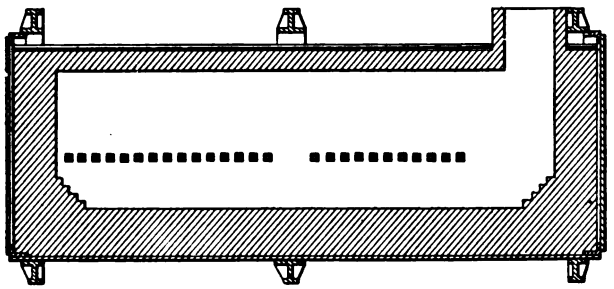


FIG. 98. — Basic-lined converter — mouth at end

the converting of leady copper mattes may run as low as 20 per cent.

An important technical point in connection with this operation is the thorough drying of the silicious ore which is added to flux the iron of the matte. The cost thus far has been nearly 30 cts. per ton of blister copper, but it is expected that this will be halved

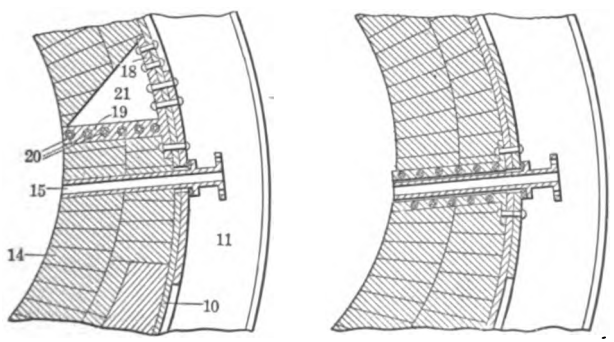


FIG. 99. — Basic-lined converter. Brackets to confine lining

by suitable driers heated by the gases from the converters. It is charged in "boats" — rectangular boxes of sheet iron, reduced at one end to a pouring-spout, and holding about 7000 lb. of ore.

Moore points out that the basic-lined converter shows greater heat losses by radiation and conduction than the ordinary vessel,

due to the thinner lining and the greater heat-conductivity of the magnesian brick. It is, therefore, necessary to work with larger charges of lower-grade mattes, and to ensure greater thermal activity by using a larger volume of blast. The temperature is generally lower than in the acid-lined converter, and it is necessary to keep two men punching tuyeres continuously.

While running only one converter at Garfield, there was an enormous proportion of lost time. Moore states that in blowing matte for the production of 1248 tons of copper bullion, the actual blowing time was only 12½ hours out of the twenty-four. The

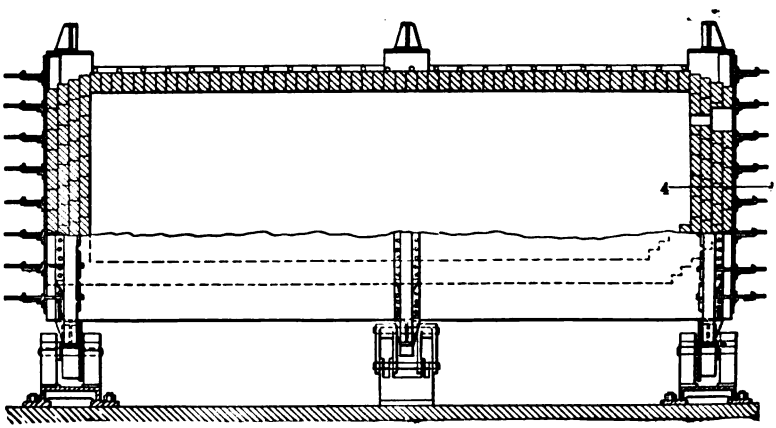


FIG. 100. — Basic-lined converter with telescopic ends — side sectional elevation

rate of production was 42 tons of blister copper per day, from mattes averaging 43 per cent copper.

The lost time is accounted for as follows:

TIME LOST DURING TWENTY-FOUR HOURS

Ladling copper	3½	hours
Forming initial bath	3	"
Skimming slag	3	"
Charging silicious ores, and waiting for matte	2	"
Total	11½	hours

Much of this loss will be remedied when this new system is once properly established. Ladling the copper bullion into the furnace for partial refining takes 1½ hours; this will be reduced

one-third by the use of a three-spout ladle. The time lost in charging silicious ores and other solid materials will be greatly reduced by the use of hoppers set over the mouth of the converter, and other improvements are under way.

I quote Moore's comparison of the basic and acid converters: "The basic-lined converter has the advantage of cheaper construction, as it requires no relining plant; less building space and lighter crane; there is no handling of vessels, hence a larger vessel may be used; no clay is used for binder in relining; less restriction on the class of silicious ores which may be used as flux; less slag is produced; lower-grade matte can be successfully converted

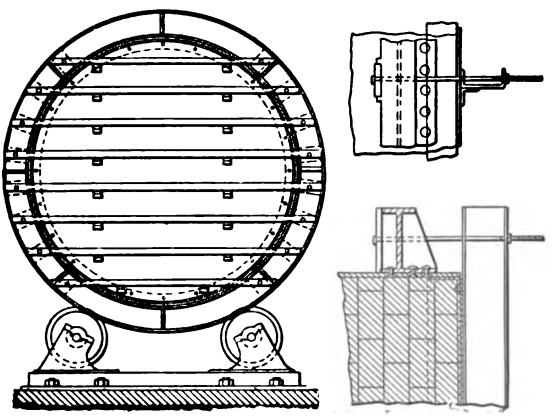


FIG. 101. — Basic-lined converter — section of head and details of cramp-bars

at a less cost per ton of matte and silicious ore than the same concentration can be effected in the blast-furnace.

"The disadvantages are: losses due to blowing out fines from the dried silicious ores; lost time while relining and repairing; extra careful manipulation required to avoid premature burning out of lining and tuyeres, or the formation of accretions; extra tuyere-punching required on account of the lower temperature. The cost of relining and drying acid converter shells may, under local conditions, be much less than the cost of repairing the lining, and drying the silicious ores.

"Summing up these advantages and disadvantages, it will be seen that there are many localities where there will be a distinct disadvantage in installing the basic-lined converter, and many

where its advantages would produce an ultimate plant profit. For example: Anaconda, with clayey concentrator slimes for a binder and suitable quartzose ores for a flux, if operating the converters as carefully as the basic converters have been operated, would probably show no advantage with a basic-lined plant; while Garfield, with barren clays and practically no margins on silicious ores suitable for linings, would find a great gain in using basic linings, even if the acid-lined plant were operated efficiently. At Aguas Calientes there is a good silicious ore and cheap labor, but barren clays are being used; here the silicious lining would seem to be the cheaper, since the cost of relining is less than the cost of drying silicious ore and replacing tuyeres and magnesite brick.

"There are, however, many places where the basic lining will prove advantageous, especially where low-grade mattes are produced."

Apparently, the chief features which have made this vessel successful where other magnesite converters have failed are the great size of the vessel and the retention of the heat by conserving the hot gases more carefully than in the ordinary converter; to this must be added the patience and skill of Pierce and Smith, and the persistence of the company at whose plant the apparatus was perfected.

As may be gathered from the preceding description, the basic-lined converter does not offer a radically novel means of converting mattes at a low cost. It gives us, rather, a parallel method which may suit admirably certain local conditions that are not favorable for the employment of an acid lining.

The average output per 24 hours, at Garfield, for one converter running on 42 per cent matte, has been about 66,000 lb. blister copper.

The converter requires 6000 to 8000 cu. ft. air per minute (engine displacement), using a pressure of 10 to 12 lb. per square inch.

The actual labor at the vessel requires three men per shift — a skimmer and two punchers.

The repairs consist mainly in replacing tuyeres and magnesite brick at the horizon of the tuyeres, and their extent and frequency depend mainly upon the care and skill employed in managing the operation. The amount of blister copper produced before the

vessel requires radical repairs in the tuyere region has varied between 700 and 2800 tons, the cost of relining thus ranging from 12 cts. to 50 cts. per ton blister. These repairs usually cost from $250 to $350, and require a shutdown of four to six days, half of which time goes to the cooling of the vessel sufficiently to permit working in its interior.

The actual cost of replacing an acid converter with one of these large basic-lined converters, in the western mining states, has been in the neighborhood of $20,000. When running on a 40 per cent matte, its daily duty will not vary much from that of one of the full-sized ordinary acid-lined vessels.

It is worked on a large scale, and with satisfactory results, in converting the impure leady copper mattes resulting from silver-lead smelting and, in connection with a bag-house, has shown no unusual metallurgical losses of silver or lead.

At the Washoe smelter several magnesite-lined converters are now in operation with satisfactory results, the old converter shells being simply provided with a basic lining. The management does not wish to publish details until the experimental stage is passed, but I am informed that very little difficulty has been experienced, and that — except for the lining — no change has been made in the vessels excepting to increase the diameter of the tuyeres to $1\frac{1}{2}$ inches. About 25 per cent of silicious ore is added to flux the iron of the matte. The main point in managing the blow seems to be to prevent any excessive heat, which destroys the magnesite lining rapidly.

The Knudsen method of smelting pyritic copper ores in a modified converter has been described fully by its originator in *The Mineral Industry*, Vol. XVII, page 315.

This is a process which, under the management and development of the inventor, has been at operation at the copper mines of Sulitjelma, Norway, since the year 1903, and has attained sufficient commercial success to warrant its admission to the list of methods that have passed the experimental stage.

The process is, virtually, a series of disconnected operations, or cycles, each cycle occupying about $4\frac{1}{2}$ hours and comprising two stages — pyrite smelting and bessemerizing. It is carried out in a special type of converter, lined with magnesite brick, and with tuyeres only an inch or two above the bottom, the vessel

being greatly narrowed in that region. Each fresh charge of raw ore is kindled with a small amount of coke, and the furnace, under the influence of a strong blast, soon begins to act as a true pyrite smelter, the sulphides liquating from the superincumbent mass, and collecting in the narrowed reservoir at the tuyeres. As soon as the tuyeres are covered, the blast is increased, and is now blowing through a bath of liquid matte instead of through a shower of molten sulphide globules. In other words, bessemerizing has now begun, and pyrite smelting has ceased, unless — as is quite probable — the shallow layer of matte does not absorb all of the oxygen of the blast, and a certain amount of pyrite smelting continues simultaneously with the bessemerizing.

Fresh accessions of liquated sulphides still keep the grade of the matte low; but within two hours this supply ceases, and the actual concentration of the matte begins, the superincumbent silicious remains of the ore-charge being fluxed gradually into a liquid slag as the formation of ferrous oxide proceeds. The rapid oxidation of iron and sulphur, due to a powerful blast acting upon a highly ferruginous matte, produces an unusually high temperature, and it is just at this point that the condition of the operation becomes quite different from what it would be in the pyrite furnace. In the latter apparatus the slag escapes from the focus as soon as it is formed, while in the Knudsen vessel the slag is retained for a long time in contact with the slowly melting silicious skeleton which represents the least fusible portion of the original ore.

We know already that — nothing hindering — the higher the temperature, the higher in iron (and lower in silica) will be the resulting slag, because the formation-temperature of the more ferruginous silicates is higher than that of the more acid ones. Consequently, under the conditions obtaining in the Knudsen converter, we should expect a slag particularly high in iron, and low in silica. As a matter of fact, however, Knudsen is able to produce an unusually silicious slag when desired, running as high as 40 per cent silica and 10 per cent alumina. This comes, manifestly, from the long contact of the originally formed ferrous unisilicate with the silicious and aluminous gangue at a high temperature.

While able to blow the matte up to blister copper, the apparatus is, of course, not designed for that purpose, the inventor preferring

to stop blowing when the matte contains about 50 per cent copper. This makes an advantageous product for ordinary bessemerizing, and also yields a slag sufficiently poor to be thrown away.

As might be expected, the mere quiescence of the melted charge in the converter at the termination of the blow is not enough to settle the matte out of the slag properly. An external, separately heated, forehearth or settler seems to be an essential part of the plant.

While giving all possible credit to Knudsen for his successful solution of a difficult mechanical problem, I cannot see that his process is likely to be of advantage to large operators who have an abundant ore supply and ample capital to construct any type or size of apparatus that economical working may demand. Speaking from the standpoint of the large operators, any step from continuity to periodicity is a step backward, especially in this country where the trade unions seem determined to force the elimination of human intervention from all industrial processes.

Why a small pyrite furnace, stopping every 4½ hours to blow-out and blow-in, is better than a large pyrite furnace running continuously, I cannot see.

I can see very clearly, however, that if the results that can actually be attained with this little apparatus by others equal those reported by the inventor, and especially if the amount of repairs can be kept down to his standard (for instance, one magnesite brick per ton of ore treated), it may prove very valuable to the small operator, especially in isolated districts.[1]

"The improvements that have been made in copper converting during the past thirty years may be tabulated as follows:

"(1) Enlarging the size of the converter vessel has effected higher temperatures through the greater blast volume used, more regular and easier operation and less losses through the production of 'slop' and cleanings that must be resmelted.

[1] My personal experience in the converting of copper mattes does not include the late improvements and, instead of attempting to sum up the situation in my own words, I feel that it will be more useful to quote the opinions of an engineer who is still actively engaged in this class of work. In the *Eng. and Mining Journal* of Sept. 3, 1910, there is an excellent paper by Redick R. Moore on "Recent Practice in Copper Matte Converting," from which I will quote several detached paragraphs.

"(2) Enlarging settlers of the blast-furnaces has furnished storage for matte and avoided the cost of resmelting matte for the converters in special furnaces.

"(3) Putting in the linings with power rammers and using gold, silver, and copper ores in the place of barren quartz and clay.

"(4) Improved mechanical appliances for tilting converters and machinery for handling matte, slag, blister copper, and other products.

"(5) Improvements in means of getting the air blast into the matte and preventing leakage.

"(6) The introduction of a basic lining in place of the silicious lining, effecting a longer life of lining and avoiding the cost of an expensive relining plant and cranes of great tonnage for handling heavy converters; utilization of silicious ores that would be entirely unsuitable for lining acid-lined converters to flux the iron of the matte.

"The improvements that have been made, due to the enlargement of the converting vessel, have been limited, in the acid-lined converter, by the mechanical means that have been installed for handling the vessels and relining them. Owing to this limitation, little change has been made in the last few years in the size of the converter — the largest vertical converters, 8 ft. in diameter and 16 ft. high, have been in use at Aguas Calientes for over 15 years; and the largest acid-lined horizontal converters, 8 ft. in diameter by 12½ ft. long, have been in use for over six years. The more recent installations have been mostly of smaller sizes, 7 ft. by 14 ft. and 8 ft. by 11½ ft. respectively, or smaller, although the International plant at Tooele, Utah, is installing 8 by 12½ ft. barrel converters which would indicate that size to be entirely satisfactory at Anaconda.

"The large-sized settlers of round or elliptical section that were made necessary to avoid resmelting mattes have proved a considerable improvement to blast-furnace practice as well as to converting, and no one would consider erecting a large copper-blast furnace now without a large settler whether converters were to be used or not.

"The advantage of thorough tamping of the converter lining was appreciated some years ago and adopted generally. The usual appliance in use is an air-hammer hung on a traveling carrier from a movable jib-crane arm.

"The use of the silicious ores in the place of quartz or quartzite has made a market for ores that would otherwise not pay the cost of smelting. The use of clayey ores and tailing slimes has introduced quite a saving as compared to barren clay, but the supply of clayey ores is now insufficient in Mexico and barren clays are being used extensively, as tailing slimes do not always contain the binding qualities that are necessary. Attempts to use the slimes from the Utah Copper mill were unsuccessful at Garfield.

"It is important in putting in acid linings that it should be correctly tempered with just the proper amount of moisture to produce a hard homogeneous mass when tamped. Making a fixed number of linings per shift, or putting out relined converters regardless of the manner and thoroughness with which they are tamped, results invariably in short life and more expensive relining in the end. When correctly mixed and rammed the lining should require but little drying.

"Excessive amounts of fines in the silica make it unfit for lining, and considerable amounts of iron oxides or lime greatly reduce the life of the lining on account of their fluxing action. For this same reason the clay used for a binder should not contain much of such fluxes, as soda, potash, iron or manganese oxides, or lime. A clay otherwise suitable, but containing as high as 5 per cent and 6 per cent CaO as gypsum, was used at Garfield, but did not give a satisfactory life of lining.

"To be entirely suitable for lining acid converters, the clay should be refractory enough to hold the particles of silica until they are entirely fluxed by the iron oxide formed in the bath of matte. Otherwise pieces of silica are loosened from the lining and floated out with the slag unfluxed.

"A great many improvements have been made in machinery for tilting devices, but the arrangement utilized on the Pierce and Smith design of basic-lined converter is probably the simplest and most economical. This consists of two wire ropes, one end attached to the converter with the other end of each attached to the heads of the plungers of a hydraulic cylinder, set on opposite sides of the converter. These avoid the use of the rack and sector and give a very steady and easily regulated motion to the converter. Of course, either alternating- or direct-current motors could be used in place of the hydraulic cylinders if preferred, or in a cold climate where the use of water is likely to give trouble from freezing.

"For handling matte and shells for relining, the traveling electric crane is in general use and in most recent plans the furnaces and converters are arranged for the handling of the matte from the settler directly to the converters by the crane. In another system in use at Aguas Calientes since the converters were installed, there are no cranes; the matte is handled from the settlers to hydraulic platform elevators, each serving two stands, by ladles set on trucks with worm gears for tilting. The ladles are hauled by steam locomotives, and the contents poured into the converters through a movable, curved, clay-lined spout. The shells are handled into and out of the stand by other hydraulic platform elevators, constructed directly underneath the stands on special tracks built for the purpose. They are handled by hand-car movers to a transfer table and in the same manner to any desired relining stand.

"At Anaconda, where the converter building is separate and removed from the furnace buildings, the matte is hauled from the furnaces to the converters by locomotives with ladles set upon trucks and there poured direct from the trucks to the converters through short launders.

"Without careful analysis and complete understanding of conditions anyone would be inclined to label the first methods modern, up-to-date, and economical; the second, out-of-date, crude and expensive, and the third, double handling; in fact, I have heard these identical adjectives applied to the several methods by metallurgists who had not operated any of them, or only one of them, and had not taken special conditions into account.

"The special disadvantages of the first method as compared to the second are: (1) Higher first cost; (2) Greater repairs and upkeep charges; (3) Breakages; (4) Danger from heavy loads carried over head and demoralization of operators of converters; (5) Inefficiency of crane operators from smoke, fumes, and explosions; (6) Inability to operate furnaces efficiently in an atmosphere heavily charged with converter gases.

"It may be said that it is always cheaper to transport on the ground level than by any means that move small loads in the air overhead, and the repairs to motor or engine, cars and tracks are also much less. At Aguas Calientes the cost per ton of copper for delivering matte to the converters and handling slag from them to the slag beds was only a small fraction of the cost of the same

operations at Garfield, although the output was only one-half or thereabouts.

"The method at Aguas Calientes had several obvious disadvantages that could readily be avoided in building a new plant. It should also be taken into account that they have cheap and efficient labor when intelligently handled. Where there is sufficient difference in levels the hydraulic platform hoists for hoisting the matte cars could be avoided and the matte poured into the converters from the matte-tap level by short launders. The hydraulic hoists for changing converters, which are an ever-present source of danger, could be replaced by a heavy short-span crane for handling converters to relining stands. A crane for handling heavy material is practically a necessity in any case.

"The suffocating gases from the converters while pouring copper, turning up and down, etc., are abundant reasons for placing the converters at a distance from the furnaces and the difference in efficiency and 'morale' of the men employed on the furnaces should much more than pay the cost of the transportation and handling. High buildings are not a panacea for smoke and fume troubles, especially if they are left open all around the bottom so that the wind can blow through and cause enough suction to neutralize any draft that would otherwise be obtained. Working in an atmosphere of sulphurous-acid gas for eight hours, it is to be expected that the cranemen will be far from efficient and that the breakage and accidents will be excessive.

Handling the Blister Copper

"For handling the blister copper, bullion furnaces and tilting ladles with casting machines have been installed in a number of the larger plants. Such installations allow for the prompt release of the converter when its charge is finished and, when properly designed and operated, should make for cheaper handling and loading of the blister copper and a cleaner product for shipment. Smaller plants use the system of pouring into molds, placed on trucks and moved by hand or by the motive power used for handling the slag. By setting a three-spout ladle under the launder that is used to break the fall of the stream of copper, three molds may be filled at a time.

"The molds for casting the copper are a considerable source of cost and annoyance. They are usually made of cast iron, although

copper molds with a plate of cast iron in the bottom where the stream strikes have given good satisfaction. To allow accurate sampling at the refinery the copper must be cast into thin plates. A thickness of two to three inches is generally acceptable. Wraith's experiments[1] sampling anode copper at Anaconda confirm this."

Any study of the elimination of the obstinate impurities (Bi, As, Sb, Te, Se) during the bessemerizing operation would occupy more space than can be afforded to any such subject in these pages.[2]

Bismuth is removed in a highly satisfactory manner by the converter, being driven off mainly by volatilization. Experiments show that 94 to 96 per cent of this injurious element is removed in this manner. Arsenic is volatilized to the extent of from 70 to 90 per cent, one-half of it, or more, being driven off during the first 30 minutes of the blow. Antimony is less satisfactory, losing 60 to 73 per cent. Selenium and tellurium are removed to about the same extent as antimony.

It is interesting to learn from Mathewson's experiments that, when blowing a tolerably pure 46 per cent matte, 96 per cent of the iron and 53 per cent of the sulphur were removed during the first, or slagging period.

[1] *Bull.* A. I. M. E., March, 1910.

[2] Among many good papers, the following are indispensable to the student of this subject: "A Study of the Elimination of Impurities from Copper Mattes in the Reverberatory and the Converter." Edward Keller, *Trans. Am. Inst. Mng. Engrs.*, Vol. XXVIII, p. 127. "The Elimination of Arsenic, Antimony, and Bismuth from Copper." Allan Gibb, *Trans. Am. Inst. Mng. Engrs.*, Vol. XXXIII, p. 653. "Relative Elimination of Iron, Sulphur, and Arsenic in Bessemerizing Copper Mattes." E. P. Mathewson, *Trans. Am. Inst. Mng. Engrs.*, Vol. XXXVIII, p. 154.

CHAPTER XV

THE REFINING OF COPPER

It is unfortunate that the term "refining" which the copper smelter has applied for generations to a specific and highly specialized furnace-operation should also have been universally adopted, in recent years, as a portion of the name of an extremely modern process whereby impure metallic copper, produced from previous metallurgical operations, is dissolved in sulphuric acid and precipitated in a condition of comparative purity by the aid of electricity. This process, of course, bears the name of "electrolytic refining," and does not fit the copper for the market. In this book the term "refining" will be applied exclusively to the final furnace-operation by which the copper is fitted for industrial purposes.

This terminal refining operation is like a lake which has many inlets, but only one outlet. Its outlet is the market, while its sources of supply are various: converter bars from the bessemer process; blister copper from the Welsh blister process; cathodes from the electrolytic refinery; black copper from the smelting of carbonate ores in the blast-furnace; cement copper from one of the wet methods; scrap copper purchased in the market, or from processes about the smelter; and other less usual materials.

This terminal process is a very delicate operation and cannot cope economically with any considerable amount of the more harmful impurities which the metallic copper is likely to contain. It is found more advantageous to remove the great bulk of the impurities by more suitable means, and to send to the refining furnace metal that is already nearly pure, containing usually 98 per cent, or more, of copper. Thus the refining furnace operation can scarcely be considered as a process practised for the removal of impurities, but rather as means of imparting to the copper the last delicate finishing touches which it requires to enable it to develop its highest qualities, and to mould it into forms suitable for the purposes for which it is intended.

Electrolytic refining is a specialty and has now developed to such an extent that I am unable to include it in this volume. A thorough comprehension, however, of the results effected by this operation, and of the reasons which influence us in submitting certain coppers to electrolytic refining, or in withholding others, is essential to every miner and smelter of this metal, and demands a brief review at this point.

As already stated, in this process we dissolve the impure metallic copper — received from the smelter in the form of large plates or slabs, known as anode-plates — in sulphuric acid, and precipitate it, by means of electricity, as practically pure metallic copper in the shape of cathode plates. These cathodes usually require no further purification for the market, the copper being already in as pure a condition as we can expect to obtain it. It is, however, necessary to give it a final melting in order to mould it into commercial shapes, and to obtain it in a more dense and homogeneous condition than it possesses in the cathode. Thus it will be seen that furnace-refining is always the terminal operation, even for the pure copper produced by electrolytic refining.

As it has just been stated that the electrolytic refiner receives copper containing impurities, and delivers the metal as pure copper, it is plain that the purpose of the process is to separate the impurities from the copper. This is quite true; yet there is an important distinction to make between certain of the impurities themselves; for instance, antimony is a common and most injurious impurity in the metallic copper resulting from certain mines. The same copper, very likely, also contains gold, which is also an impurity; but gold is not only harmless in its effect upon the copper, but is enormously valuable in itself. Both of these impurities — as well as all others — are separated from the copper in the process we are now discussing. The question, therefore, might arise whether electrolytic refining is practised to remove *harmful* impurities from the copper, or to extract its *valuable* impurities, such as gold, silver, and platinum.

As a matter of fact, it is used for either purpose or, as is more common, for both purposes simultaneously. Hence we might classify all the varieties of copper which the electrolytic refiner treats under three heads:

(*a*) Copper containing harmful impurities, but without any

of the precious metals. Submitted to electrolytic refining solely for purposes of purification.

(*b*) Copper free from harmful impurities, but containing sufficient of the precious metals to pay the expense of this process and leave a profit. Submitted to electrolytic refining solely for the purpose of separating the precious metals. (This is a rare group, owing to the fact that gold and silver are usually accompanied by certain peculiarly harmful impurities.)

(*c*) Copper containing harmful impurities, and also the precious metals. Submitted to electrolytic refining for the double purpose of removing the impurities as well as obtaining the precious metals. (This is the most common group.)

It is a common, but misleading, statement to say that "the electrolytic process separates the gold and silver from the copper." This would seem to imply that the process had some selective effect upon the gold and silver, and rescued them from the grasp of the copper; whereas, as a matter of fact, the action of the process is simply to dissolve and precipitate pure copper, leaving all other substances (impurities) behind; and if a portion of these impurities happens to consist of gold and silver, — so much the better for the owner of the copper.

The working up of these residual impurities (slimes) belongs strictly to the operation of electrolytic refining, and does not concern us in this volume.

Before leaving the subject, however, I must call attention to one more point which is of great practical importance to the copper smelter.

The process of electrolytic refining is of such a nature that it cannot well be connected with an ordinary smelter. It requires cheap power, highly trained chemists and electricians, and a general scale of operations too large to be in harmony with any but an exceptional smelting plant. It has thus come about that this branch of metallurgy has become centralized in a few great custom refineries which purchase the impure metallic copper from the smelters, or charge a fixed rate per ton for treating it. As their tariff is based, to a considerable extent, upon the amount of impurities contained in the copper sent from the smelters, it is of prime importance to the latter to be able to determine just how far it pays them to go in purifying their copper before shipping it to the electrolytic refinery. This involves some knowledge

of the effect of impurities upon copper, of the possibility and rate of their elimination by furnace processes, and of the most suitable methods for effecting this partial (rarely complete) elimination.[1]

The more important qualities which copper must possess before it can be marketed at a suitable price are malleability, ductility, and electrical conductivity — combined with strength. Pure copper possesses all of these qualities to an unusual degree, but it is an unfortunate circumstance that some one, or all, of them may be diminished to an extraordinary extent by the presence in the copper of comparatively minute quantities of certain other elements which occur frequently in company with the ores of copper.

Many of the foreign substances which would damage the copper are so volatile, or so oxidizable, that they are vaporized or slagged in the various furnace processes to which the ore is subjected before the final impure metallic copper is produced; but five of the worst of them cling to the copper with wonderful tenacity through the operations of roasting, smelting, and converting, and although their proportion lessens in each of these operations, there is still enough of them left to interfere seriously with the resulting metallic copper. As furnace operations are manifestly incapable of coping satisfactorily with these five impurities — antimony, bismuth, arsenic, selenium, and tellurium — it becomes evident that when we know of the presence of any one, or more, of these in our original ores, we may feel pretty sure that our impure metallic copper cannot be made into a marketable commodity by the simple process of furnace-refining, but that the furnace-refining must be preceded by the operation of electrolytic refining.

As already indicated, however, even the electrolytic process cannot advantageously handle impure coppers containing too much of these, or any other, impurities. They dissolve slightly in the acid bath and make it so impure that the process is hampered greatly. Hence, we see that we must take care to produce a reasonably high-grade metallic copper from our furnace operations, in order that the electrolytic refiner may not be too much hampered by impurities. This particular point has no direct bearing upon the furnace-refining of copper; but it is impossible

[1] See "Principles of Copper Smelting," chapters XI and XII, for a more detailed study of this subject.

to have a clear understanding of the final operation without having some general idea of the intermediate steps.

Returning now to the furnace-refining proper, we recognize that metallic copper, to be suitable for this operation, must not contain any appreciable amount of the five fatal substances, nor any such amount of the common easily-removable substances (such as iron, sulphur, lead, etc.) as will protract the refining to an unreasonable length of time. The refining furnace demands highly-paid labor and supervision, and we desire usually to work off a charge every 24 hours. This could not be done if we started with a copper containing too much iron, sulphur, nickel, lead, etc., as the time required for oxidizing and slagging, or volatilizing, them would be excessive. Consequently, the refiner demands a copper containing seldom more than two per cent of foreign substances, and free from any appreciable quantity of arsenic, antimony, bismuth, selenium, or tellurium. If the impure metallic copper produced at the smelter fulfils these requirements (and does not contain profitable amounts of the precious metals) it may pass directly to the refining furnace without any necessity for a previous electrolytic operation. In such case a good-sized smelter might very probably do its own refining, unless high local expenses, and the cheaper railway freights on bars of unrefined copper, made it advantageous to locate the refinery nearer a center of distribution.[1]

[1] The reader inclined to criticism may point out that my statement of the customary high tenor (98 per cent and upwards) in copper of the material submitted to furnace-refining is too sweeping; that at Lake Superior, the so-called "mineral" from the concentrating mills is subjected at once to the refining operation, and seldom contains more than 65 to 75 per cent copper, while cement copper and scrap copper of equally low grade are often added to the refining charge. These exceptions are more apparent than real. For instance: the entire refining-furnace charge often consists of Lake Superior "mineral" containing, perhaps, only 65 per cent copper. Yet the 35 per cent of foreign material consists solely of particles of metallic iron from the stamps, fragments of hematite or magnetite, and similar heavy minerals. The little grains of metallic copper which are present are already exquisitely pure, and consist of over 99 per cent of copper, with no appreciable admixture of any of the five peculiarly injurious elements. The 35 per cent of foreign material is not in any way alloyed with, or dissolved in, the copper and, as soon as the whole mass is melted, will float on the top as slag, leaving the 99 per cent pure copper perfectly free to undergo its refining.

The same is true of some cement coppers, the foreign admixture being

For convenience of description, the furnace-refining of copper may be divided into two principal stages: (*a*) stage of oxidation; (*b*) stage of reduction.

The purpose of submitting the melted copper to these two consecutive, opposing influences is first to oxidize the minute proportion of foreign substances still present, and encourage them to form a slag which floats upon the surface of the bath. In order, however, to effect this purpose thoroughly, it is necessary to carry the oxidizing influence to an extreme, and to oxidize some four or five per cent of the entire copper. A portion of this oxidized copper behaves like the oxides of the foreign substances which we desire to remove: that is to say, it combines with silica from the lining, forms a slag on the surface of the bath, and is skimmed off, being thus lost to the operation and requiring subsequent treatment for its recovery.

Another portion, however, is dissolved by the melted copper in the form of cuprous oxide and, aided by the vigorous boiling of the copper, permeates the entire bath, carrying oxygen to the more oxidizable foreign elements and being itself reduced back to metallic copper. This oxidizing process usually has to be continued until the bath contains some six per cent of cuprous oxide. This is as far as it is beneficial to push it, and the reduction of the cuprous oxide to metallic copper is begun, by means of the "poling" which will be described later.

The poling is continued until the proportion of cuprous oxide is reduced to a point at which the copper shows its best physical qualities, as determined every few moments by tests upon the spot. When this exact point has been reached, the copper is held in this condition, without either oxidation or reduction, until it has all been cast into the desired shapes.

The furnace is then examined, prepared for the next charge, and filled with the proper amount of impure copper, the entire cycle of operations recurring every 24 hours.

Until recent years the copper-refining furnace was always of

mainly oxides of iron, and various basic salts, which will float on top of the pure copper at once. Even in scrap copper, most of the foreign substances are only mixed mechanically with the fragments of copper, and the small proportion that may exist as an alloy with the copper is of such a nature that it will be rapidly volatilized or slagged; otherwise, it could not be added to the refining charge without delaying or injuring the operation.

small dimensions, as were all reverberatory furnaces. But even a comparatively small hearth can contain a great weight of melted copper, and it is probable that the size of refining charges would have increased materially, had it not been for two limiting influences. One of these was the capacity of the attendants to fill and empty the furnace, and still complete the refining of the charge within the prescribed 24 hours. The other was the fact that we were afraid to carry so large an accumulation of melted copper upon the furnace hearth. An outbreak of metallic copper is a serious occurrence. It is almost impossible to stop it until the furnace has emptied itself, and a mass of several tons of malleable copper is a difficult object to deal with.

The first of these objections has been met by the substitution of mechanical devices for hand labor; the second, by improved construction of furnace and hearth, until now there seems almost no limit to the weight of copper that may be refined in a single operation. Indeed, the only actual limitation at the present time seems to be the desirability of keeping each unit down to a size where its stoppage will not paralyze the effectiveness of the plant.

Before examining the construction of the refining furnace and the technical details of the operation, it will be convenient to follow the changes which the copper undergoes from the time it enters the furnace until it is cast into the shapes desired by the market. When we consider the comparative purity of the original impure copper subjected to the process, it is evident that the changes which it undergoes in the refining furnace must be of an exceedingly minute and subtle character, and most difficult to demonstrate with certainty. It was not until metallography came to the aid of chemistry that we began to gain a clearer comprehension of the important rôle played by mere physical changes in the constitution of the metal. As minute and elusive as these changes yet appear to us, they have an extraordinary effect upon the quality of the copper, and while the practical refiner needs neither chemistry nor metallography to enable him to develop the very highest attainable qualities in a hearthful of molten pig copper, it is only by the aid of these sciences that we can understand exactly what he is doing, or can expect to cope quickly with difficult and unusual conditions. This minute study of delicate reactions and physical changes, still far from complete, would be totally out of place in a concise practical

book, and I must confine my description to a hasty résumé of the more striking and important changes that the copper undergoes during this process.[1]

The elimination of the impurities is, of course, the subject most important to the practical refiner, and Wanjukoff's elaborate tests at the Wysk smelter, in the province of Perm, Russia, enable us to study this point more thoroughly and systematically than has ever been the case hitherto. Without giving a full translation of his remarks in connection with the elimination of the various foreign substances, I make free use of them, and also reproduce the graphic representation of his analyses.

The refining furnace in which he conducted his experiments was of small size, and was fired with gas produced from twigs, bushes, pine needles, and similar waste material. The copper to be refined was of unusually low grade, according to our standard, although, its chief impurities being iron and sulphur, this circumstance need have no deleterious effect upon the quality of the product. Arsenic is present to the extent of about one-eighth of one per cent, and would render the resulting refined copper unfit for electrical purposes, but not for rolling. The analysis of the copper to be refined was as follows:

Cu	94.55
Fe	3.0373
Ni	0.4080
Co	0.8944
Ag	0.0021
Pb	0.0123
As	0.1257
Sb	0.0020
S	0.8678
O	0.0953
P	0.0105
SO_2	0.0006
Insoluble	0.0004
	100.0064

[1] In Chapter XI, "Principles of Copper Smelting," will be found a fuller discussion of this subject, as well as references to various papers dealing with it more minutely. Two monographs which I find very useful at present are "A Laboratory Study of the Stages in the Refining of Copper," by Hofman, Green, and Yerxa, *Trans. Am. Inst. Mng. Engrs.*, XXXIV, 671, and a most scholarly monograph by Wanjukoff, in *Metallurgie*, 1909, pp. 749 to 760 and 792 to 802.

The furnace was exceedingly small, receiving only 4450 lb. (2022 kg.) copper, but was under perfect control as regards temperature and atmosphere, as it was heated by producer-gas.

The temperature determinations of the bath of metal are valuable. They were made with a Le Chatelier thermoelectric pyrometer, and gave the following results:

	Temperature in Deg. C.
end of melting-down period	1104
preceding boiling	1091
maximum boiling	1123
end of boiling	1113
poling	1113
refined copper	1102

In the following diagram Wanjukoff depicts the gradual elimination of the impurities during the successive stages of the refining operation, the horizontal lines representing the percentages, and the vertical lines, the time. For obvious reasons he finds it more convenient to adopt a different scale for certain of the impurities; this is indicated upon the diagram. As a basis for the diagram, 100 parts of copper are taken, and the analyses of all the succeeding samples are calculated upon this basis. As a small proportion of copper is removed constantly by slagging, this introduces a minute error, making the amount of the impurities seem slightly higher than is actually the case; but the important point is the relative elimination of the impurities. The dotted lines at the beginning of the curves indicate the probable rate of oxidation of the impurities from the time of charging until complete fusion. As remarked by Wanjukoff, this may be regarded as a storing-up of potential energy which becomes kinetic when the metallic oxides melt together with the quartz lining of the hearth.

Iron becomes oxidized rapidly during the melting-down period, is slagged vigorously as soon as the temperature permits its union (as FeO) with SiO_2, and, at the beginning of the boiling-period, is reduced to only 0.0081 per cent. Beyond this point its elimination falls off rapidly and soon becomes practically *nil.*

Cobalt, as might be expected from its nature, behaves in a manner somewhat similar to iron, excepting that its early oxida-

tion cannot be detected, owing to the unstability of its free oxides at a red heat. As soon as the temperature is high enough to encourage its combination with SiO_2, it is slagged rapidly. Its subsequent behavior is almost identical with that of iron.

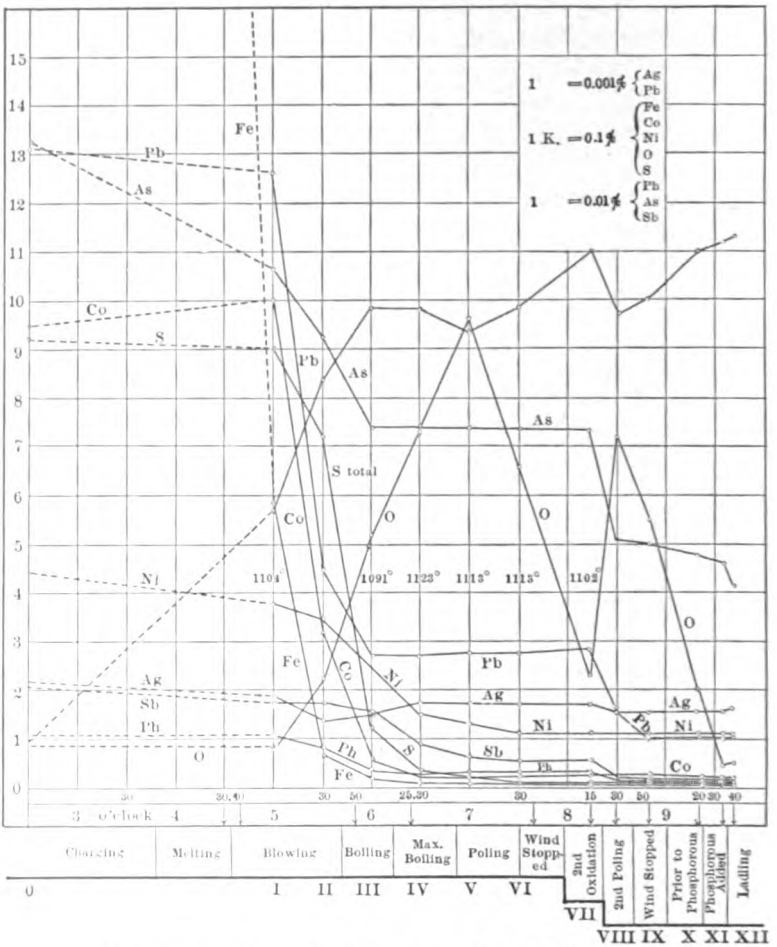


FIG. 102. — Removal of impurities during refining

Nickel behaves quite differently, refusing to take advantage of the oxidizing conditions in the hearth until both Fe and Co have diminished greatly. Its most rapid oxidation occurs a little before the period of boiling, after which it slackens rapidly.

Sulphur is burned off to some extent during the fusion of the copper, but its removal is slow so long as much iron is present. After the appearance of most of the Fe, Ni, and Co, the sulphur burns rapidly, without any increase in the oxidizing condition of the hearth. During the period of boiling, the familiar reaction between cuprous oxide and cuprous sulphide ($2Cu_2O + Cu_2S = 6Cu + SO_2$) occurs, the SO_2 dissolving in the bath of copper until saturation is produced, when it escapes rapidly, causing the well-known phenomenon of "boiling." The succeeding operation of poling not only reduces Cu_2O to Cu, but drives out the dissolved SO_2, partly by mechanical ebullition, partly by the displacing effect of CO and H.

Lead is removed mostly during the period of oxidation. The stages of boiling and poling witness but a minute change in its amount, but as soon as the second period of oxidation begins, the lead again commences to be slagged.[1]

Silver. This metal happens to be present in the copper in extremely small proportion. So far as can be observed, it follows the behavior of the lead, though in a greatly diminished degree.

Antimony oxidizes more freely during the period of fusion than during the later oxidizing stages, when protected by the slag. The period of boiling witnesses a considerable oxidation of the antimony, the rapid generation of SO_2 being a strong influence in its elimination. The Sb still remaining lies nearly dormant during the first poling, but becomes active again during the second refining (that is to say, the second oxidation, with its subsequent reduction). There still remains, however, an appreciable amount in the refined copper.

Arsenic, during the periods both of fusion and of oxidation, burns off freely, but remains unchanged during the periods of boiling and poling. The second oxidation removes still more of it, and even the final tough-poling aids a little in this respect. The final addition of phosphorus is accompanied by a sharp drop in the arsenic.

Oxygen presents, in my opinion, the most instructive and interesting features of any of the elements studied by Wanjukoff

[1] It must be noted that the especial refining process now being studied is characterised by a double set of oxidations and polings. Also that a small addition of phosphorus is made just before the copper is ladled, and that poling is begun long before the blowing ceases.

in this experiment. I take the liberty of translating his own remarks in this connection: "In order to enable us to observe the influence of the O upon the combustion of the S, the same scale for the ordinates is retained, in the construction of its curve, that was employed for the sulphur. The O-content of the bath increases as the elimination of impurities progresses, and goes on with especial rapidity after it is no longer compelled to contribute toward the oxidation of the iron. Cu_2O is formed, which dissolves in metallic copper in indefinite proportion (Heyn)."

From the inception to the culmination of the "boiling" period there is a relative diminution in the O-content of the bath, Cu_2O again being called upon to decompose the Cu_2S present — $2Cu_2O + Cu_2S = 6Cu + SO_2$.

Shortly after the maximum evolution of SO_2 the formation of Cu_2O again outstrips its decomposition, and the O increases rapidly, reaching its highest point at the end of the boiling period.

The poling causes a correspondingly rapid loss of O, due to the hydrocarbons, carbon monoxide, and other gases which reduce the Cu_2O to metallic copper.

The following 12 analyses of the bath of copper during the refining just described show the progressive elimination of the impurities and form the basis for the diagram shown in Fig. 102:

WANJUKOFF'S ANALYSES OF THE BATH OF COPPER DURING REFINING

		0	1	2	3	4	5	6
1	Cu	94.55	96.93	98.207	98.96	98.933	98.749	98.98
2	Ag	0.0021	0.0018	0.0013	0.0016	0.0018	0.0018	0.0018
3	Pb	0.0123	0.0122	0.0044	0.0027	0.0027	0.0027	0.0027
4	Fe	3.0373	0.5683	0.0743	0.0172	0.0125	0.0131	0.0094
5	Co	0.8944	1.0207	0.3134	0.0586	0.0278	0.0236	0.0234
6	Ni	0.4080	0.3648	0.3394	0.2531	0.1524	0.1377	0.1100
7	Ph	0.0105	0.0112	0.0072	0.0032	0.0032	0.0032	0.0032
8	As	0.1257	0.1031	0.0914	0.0737	0.0738	0.0736	0.0738
9	Sb	0.0200	0.0174	0.0174	0.0155	0.0090	0.0066	0.0052
10	O	0.0954	0.0837	0.2123	0.5097	0.7308	0.9562	0.6671
11	S_{SO_2}	0.0006	0.0004	0.0004	0.0059	0.0085	0.0094	Trace
12	S total	0.8679	0.8705	0.7060	0.0902	0.0375	0.0253	0.0096
	Totals	100.0242	99.9841	99.9745	99.9914	100.0930	100.0022	99.8862

		7	8	9	10	11	12
1	Cu	99.42	98.905	99.078	99.47	99.65	99.66
2	Ag	0.0018	0.0016	0.0016	0.0016	0.0016	0.0016
3	Pb	0.0028	0.0017	0.0010	0.0010	0.0010	0.0010
4	Fe	0.0107	0.0076	0.0084	0.0084	0.0081	0.0080
5	Co	0.0226	0.0220	0.0291	0.0247	0.0230	0.0243
6	Ni	0.1098	0.1082	0.1092	0.1091	0.1091	0.1091
7	Ph	0.0032	0.0015	0.0011	0.0011	0.0012	0.0012
8	As	0.0739	0.0500	0.0497	0.0475	0.0460	0.0402
9	Sb	0.0066	0.0032	0.0017	0.0017	0.0017	0.0017
10	O	0.2285	0.7222	0.5467	0.2097	0.0444	0.0486
11	Sso_2	Trace	Trace	Trace	Trace	Trace	Trace
12	S total	0.0094	0.0093	0.0093	0.0093	0.0093	0.0093
	Totals	99.8895	99.8323	99.8348	99.8841	99.8954	99.9051

From the nature of the work that we desire to accomplish in the refining furnace, it is evident that the process must comprise certain steps, which follow each other in logical sequence until the furnace is emptied of its refined copper, and the cycle is completed. The scale upon which the process of refining is conducted does not affect this cycle of operations; they are always necessary whether the charge is large or small. What it *does* affect, however, is the manner in which the furnace is constructed, and the means employed for executing these various operations.

Manifestly, a furnace which is expected to produce 400,000 pounds of refined copper every 24 hours must be built in a different manner, and equipped with different appliances, from one which turned out only 20,000 pounds during the same time. These differences, however, apply mainly to the mechanical details, and not to the process itself.

For convenience of description, the entire cycle of operations comprised in the refining of a single charge of copper, and repeated every 24 hours, may be divided into six stages:

1. Charging
2. Melting
3. Skimming
4. Oxidizing (blowing or flapping)
5. Poling
6. Casting

1. *Charging.*—Where the extent of the business warrants it, the modern tendency is to increase very greatly the amount of

copper refined in a single charge. This is particularly the case in the United States, where high cost of labor and the tyranny of trade unions make it necessary to dispense with human labor so far as practicable. For instance: the newer refining furnaces at Lake Superior turn out 200,000 pounds refined copper; the New Jersey refineries run from 150,000 to 400,000 pounds per charge; the Anaconda anode furnaces, 200,000 to 300,000 pounds.

It is impossible to charge a refining furnace with such large quantities of copper within a reasonable time, unless some quicker method is applied than the old plan of lifting the pigs of copper by hand upon a paddle, which then was pushed into the furnace, dumping the pig in the required position.

At certain refineries, conditions are such that the copper is melted elsewhere, and simply poured into the hearth of the refining furnace. At the Washoe smelter, for instance, the molten copper from the converters is poured into the anode furnace, which simply gives it a partial refining in order to obtain smooth anodes for the electrolytic treatment.

At Lake Superior it is found advantageous, at some works, to melt the so-called "mineral" (concentrates containing about 68 per cent copper in metallic grains) in a separate furnace, known as the "melting furnace." This is charged once in 24 hours with 120,000 to 140,000 pounds of the "mineral," melted, skimmed, and the copper tapped through a lined steel launder into the refining furnace. As this amount does not make a full charge for the latter, the refining furnace also does a portion of the melting-down, the mass copper, barrel-work, old moulds, scrap, etc., being charged direct, and the operation so managed that its contents shall be liquid and ready to receive the large charge from the melting furnace.

At certain of the modern refining furnaces the charge is so large that it cannot conveniently be got into the furnace at one time; hence, it is the practice to introduce about three-fourths of it, melt down partially, and then add the remaining one-fourth. For instance, at one of the New Jersey plants, where the average refining charge of cathode copper is 400,000 lb., the charging (mechanically accomplished) is divided into two periods. The first charge weighs about 300,000 lb., requiring $1\frac{1}{2}$ to $1\frac{3}{4}$ hours to introduce. This is fired upon slowly for from three to five hours, or until the charge is melted down, though not completely afloat.

The second charge of 100,000 lb. of cathode copper is now added, and the melting-down resumed until the copper is thoroughly liquid.

Where the coal used for melting the copper carries sufficient sulphur to make itself felt by delaying the stage of oxidation, it is found of distinct advantage to protect the surfaces of the cathode plates by dipping them into milk of lime before charging into the refining furnace. This practice is adopted at the Boston and Montana smelter at Great Falls, Montana.

The mechanical apparatus used as an aid in charging the furnace may consist of a pneumatic piston, provided with tongs, running upon a trolley-rail; or of an electrically-operated rigid arm which passes through the charging-door to the farthest limits of the hearth-interior, its load being mechanically pushed off as it is withdrawn.

There are various devices of this nature, without which it would be impossible to charge a large furnace in any reasonable time.

A full-sized charging machine is stated to handle some 200,000 pounds per hour, and takes the place of 17 men.

2. *Melting.* — This step has already been considered in part under the preceding section. It calls for no especial comment, the main object being to obtain the metal in liquid form, although at some plants it is found advantageous to melt somewhat slowly and thus effect a certain amount of oxidation of impurities during this stage.

Local boiling indicates that the charge is not thoroughly afloat. When the charge is liquid and has become perfectly quiet, the slag is skimmed, and the metal is ready for oxidizing; indeed, at some plants, in order to facilitate the work, air is introduced (blown through pipes) before the melting-down is entirely completed.

At another plant where converter bars are partly refined in order to cast them into smooth anode plates, the charge is 400,000 pounds, requiring about two hours to introduce, by machinery. The melting-down takes about 12 hours, slag being skimmed as soon as it forms, so that the surface of the bath is kept as clean as possible. Compressed air, through pipes submerged in the metal, is also introduced through three side-doors, as soon as sufficient of the copper is melted to form a suitable bath.

The charging apparatus consists of an overhead crane supporting a carriage running at right angles to the craneway. Suspended from the carriage is a housing mounted to turn relatively to the carriage and supporting a peel *A*, which can be moved up and down as well as transversely. The novel feature consists of a shelf *B*, which is actuated by a bar headed in a piston in the cylinder *C*, and a stop *D*, which prevents the charge from spreading when the shelf is suddenly withdrawn. The inventors claim that with this type of crane the charge will be uniformly and evenly stacked, utilizing the entire available space of the furnace intended for charge.

3. *Skimming.* — This stage of copper refining is seldom credited with the importance that it actually possesses. Skimming means removal of slag from the face of the metal bath, and the formation of slag, at this period, indicates an oxidation and separation of

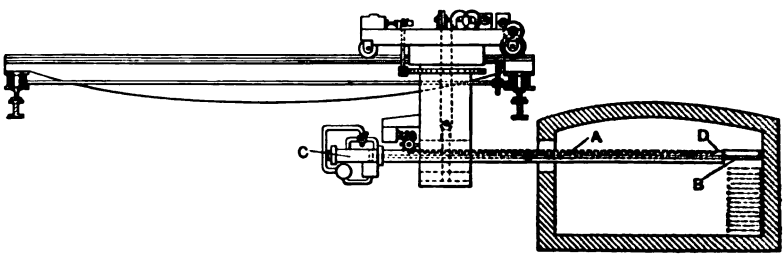


FIG. 103. — Clark and Artisell charging-crane

impurities that is an important step in the purification of the copper. A certain amount of oxidation has already taken place during the melting-down of the charge. Iron, especially, is oxidized freely during the melting period, and slags rapidly as soon as it softens and finds silica with which to combine. A certain amount of cuprous oxide also is formed, and assists in the oxidation of impurities, as soon as the bath becomes sufficiently liquid to permit its solution in the metallic copper.

There results, then, a considerable amount of slag, which continues forming until most of the iron, cobalt, nickel, lead, etc., are oxidized, floated to the surface, combined with silica, and slagged. In former days it was not the custom to begin any active measures of oxidation until the slag-forming period was closed, which occurred when the metals to which I have

just referred had been mostly slagged and removed from the furnace.

This slag may be skimmed into pots or moulds, and usually goes to a blast-furnace. It is always too rich in copper to throw away.

4. *Oxidizing.* — This is the period of vigorous concentrated oxidation, but it must not be forgotten that a considerable amount of oxidation has been taking place all through the melting-down and skimming periods, and that it has already exerted an important influence in the purification of the metal. But the oxidizing proper is not supposed to begin until after the slag-forming and skimming stages are finished, excepting in those cases, now so common with large charges, where air is blown into the bath of copper before the slag-forming and skimming are completed.

The purpose of this oxidation is, of course, to oxidize the impurities still remaining in the metal, but the manner in which it is accomplished is a very curious one. Copper, when melted, has the power of dissolving a considerable proportion of cuprous oxide (Cu_2O), and forming with it a homogeneous solution. Cuprous oxide acts as an oxidizing agent toward the impurities in the copper which we have been studying. That is to say, in the melted bath, the cuprous oxide, when brought in contact with the metallic impurity, oxidizes the latter, being reduced itself to metallic copper.

Taking advantage of these two facts, the copper refiner proceeds to form cuprous oxide as rapidly as possible, and continues doing so until a large amount — usually in the neighborhood of six per cent — has been produced, and is in solution in the metallic bath. The oxidation of such impurities as still remain progresses rapidly, and the reaction of the cuprous oxide upon still existing cuprous sulphide causes a strong development of SO_2 gas, which is accompanied by a boiling of the bath. At the completion of this stage the melted copper holds a considerable amount of SO_2 in solution, as well as a large quantity of cuprous oxide.

A button-sample of the metal will show a concave surface and, when broken, shows a single bubble at the apex of this depression. The fractured edges will show a dull brick-red color, and the metal will be weak and useless. It is now known to the refiner as "set copper," or "dry copper."

The method formerly employed for hurrying this process of

oxidation was to splash the surface of the bath with the rabble in a manner to expose the greatest possible surface to the hot gases passing over the hearth, the firing being conducted in such a way as to minimize reducing gases, and free air often being admitted through the bridge wall, or through side-ports. This operation is termed "flapping" and, in spite of the large amount of severe hand-labor involved, is still in use at certain plants.

The more customary and quicker method is to oxidize by means of currents of air compressed to 15 or 20 lb. per sq. in., and introduced into the metal bath by means of iron pipes which are plunged into the liquid. The pipes burn away gradually, but their iron is oxidized quickly and does not affect the quality of the copper. Much stronger air-pressures have been used for this purpose, but there is danger of injuring the bottom from the violent action produced by these higher pressures, and the present practice seems to be to remain within very moderate limits.

If a button-sample of the copper is taken before this oxidizing stage is begun, it will, on solidifying, show a rounded convex surface, usually with sproutings of the metal issuing from it. This is called by the refiners "spewing," or "throwing a worm," and results from the escape of the dissolved SO_2 which is squeezed out during solidification. As the blowing continues, the surface of the button-sample becomes less convex, and the "worm" less pronounced. If the copper was at all "coarse" to begin with, black specks of Cu_2S will float upon the surface of the button, concentrating at its center when it solidifies. With continued oxidation, the surface of the button becomes concave instead of convex, the black specks disappear, and, by the time the copper contains five or six per cent of cuprous oxide, the sample will exhibit a decided concavity, with a single bubble at its apex, as already described. It is now "set copper," has a dull brick-red fracture, and is weak and poor.

This completes the period of oxidation, during which a perceptible proportion of the entire bath of copper has been oxidized to Cu_2O, in order that it may search out and oxidize the minute proportions of foreign elements still in solution.

5. *Poling.* — Where converter copper, or other comparatively coarse copper, is simply undergoing a partial refining to fit it for being cast into smooth anodes, the oxidation period usually

is long, often continuing for three or four hours. But where cathodes, or other high-grade coppers, are undergoing refining, it is much shorter. Whether long or short, the intention is to produce "set copper," and if the oxidizing effect is not sufficiently thorough, the copper will not maintain its uniform quality (hold its pitch) during the operation of casting. Indeed, the degree of oxidation to be attained depends largely upon the time that it will probably take to cast the charge.

Poling, as its name indicates, consists in thrusting large poles of wood into the melted copper, and chaining them in position, pushing them in further as their extremities burn away. The object of poling is two-fold. It causes a violent boiling and splashing of the metal, by which the dissolved sulphur dioxide gas is driven out thoroughly, and it also acts to some extent in reducing the cuprous oxide present.

The pole undergoes a species of dry distillation, generating steam, hydrocarbons, carbon monoxide, and other gaseous products, which cause a profound agitation of the metal. The Germans divide the operation of poling into two periods: the Dichtpolen — or dense poling — the main purpose of which is to drive out the dissolved sulphur dioxide gas, and the subsequent Zähpolen — tough poling — the chief design of which is to reduce the dissolved cuprous oxide. This division applies rather to the somewhat coarse (impure) coppers from blast-furnace work than to the exquisitely pure cathode copper, or the high-grade blister copper, of our American refineries, and we may regard the poling as a single operation, although it is no doubt true that ebullition is the more important agent in its earlier stages and reduction in its later.

Poling cools the bath distinctly, as may easily be conceived when one reflects upon its action. The amount of heat added to the metal by the combustion of the wood itself is trifling. It probably takes as much heat to gasify the products as is gained from the combustion of these gases above the surface of the metal, and in close proximity to the flue-end of the furnace. The metal is cooled materially by the boiling and splashing, while the reduction of cuprous oxide to copper also steals heat.

The second and final stage of the poling — with impure coppers, sometimes preceded by a short period of oxidation — is

mainly a reducing process, the object being to diminish the quantity of cuprous oxide which is dissolved in the bath to a point at which the best qualities of the metal assert themselves.[1]

This point is not determined by chemical examination as to the amount of cuprous oxide still remaining in the copper; indeed, such means would be impracticable, as, at this stage of the process, the physical qualities of the copper change with astonishing rapidity, and two or three minutes of poling effect a difference in the commercial qualities of the metal that seems totally incompatible with the trifling variation in its chemical make-up that has occurred during this short interval.[2]

At some stage during the poling, the surface of the bath is covered with charcoal or clean coke to assist the reduction. This varies greatly at different plants. At some works, especially where pure cathode copper is being refined, this protecting layer is thrown into the furnace at the very beginning of the poling; at others, after the poling is partly completed, and most of the SO_2 gas has been driven out of the metal; at others, again, not until the very end of the poling.

At one of the large Eastern refineries, for instance, the bath of "set copper," resulting from the fusing and oxidizing of 200,000 lb. of cathode copper, is covered at once with 1500 lb. of clean coke of about egg size. Two poles at once are usually put into the larger refining furnaces, and 45 to 50 poles are used up, the poling requiring 2 to 3 hours. The poles are usually of poplar, spruce, or birch, and are about 25 ft. long, six inches in diameter at the butt, and three inches at the small end.

In the partial refining of blister copper for casting into anodes, the poling is, of course, a much shorter and simpler matter, as

[1] In this country, where it is rare to submit any but high-grade coppers to the process of furnace-refining, the poling is nearly always regarded as forming only a single stage; yet, in reality, the same phenomena and furnace-reactions occur as in the double-stage process just described — but in a much less striking form.

[2] Hofman, Green, and Yerxa find that, between certain limits, the percentage of cuprous oxide in copper may be determined by the microscopic examination of a polished surface of the button-sample, the whole operation requiring six to eight minutes. They suggest that this method might be useful in controlling the slight variations in pitch required for the finished products, such as ingot, cake, wire-bar, electrode copper, etc. *Trans. Am. Inst. Mng. Engrs.*, Vol. XXXIV, p. 671.

it is only necessary to continue it long enough to get a reasonably flat button, the copper approaching the condition of "tough pitch," but not yet developing the fine qualities required for merchantable copper.

Returning to the more delicate refining required for copper which is to be marketed, we find that, as the poling continues and the proportion of cuprous oxide in the metal begins to approach its normal limit, the button-samples show less depression in the center, the columnar structure of the fracture becomes more cubical, and the disagreeable brick-red color begins to shade into violet. The physical qualities of the metal improve rapidly, the successive samples — taken every few moments — show less depression and finally remain flat on cooling, and the condition of "tough-pitch" is reached. It is at this point that the metal exhibits its most admirable qualities.

The fracture of the successive button-samples during this finishing stage of refining is interesting and beautiful. The cubical structure, already referred to, changes back again into a finer columnar texture, which is, however, only the prelude to a fibrous condition, becoming eventually completely silky. Accompanying these marked changes in texture, the color changes from violet to dull red, yellowish red, and, finally, to a most beautiful rose pink.

The metal has now attained the highest physical qualities possible under these conditions, and is ready for casting at once, the sole object now being to remove it from the furnace while it is still in this admirable condition, and before it undergoes any change. Much judgment is required on the part of the refiner to "hold" it in this exact condition, permitting neither oxidation nor reduction to take place, or knowing how to neutralize either influence should one or the other, unfortunately, gain a momentary ascendancy.

The chemical analysis of various good merchantable coppers will show that, so far from being pure copper, the samples will all contain a considerable amount of cuprous oxide, varying between 0.4 and 1.2 per cent, and averaging about 0.7 per cent. Any attempt to reduce this proportion of cuprous oxide below the point indicated (varying, as just noted, for different kinds of copper) will usually result in serious impairment of the usefulness of the metal. Before such over-poled copper can be

fitted for the market, it must be oxidized, and then poled back again to tough pitch.[1]

6. *Casting.*[2] — "From the establishment of the systematic smelting of copper in Wales, about the middle of the 16th century, until the year 1895, refined copper was cast by a very laborious hand method, the workmen dipping the copper from the furnace in small ladles, which they carried to the moulds into which it was to be poured. Within 40 years, 10,000 lb. was a normal charge, and 20,000 lb. a very large one, and four to five expert ladlers were employed in dipping it. The increased scale of smelting operations demanded a similar development of the refining process, and charges were raised to about 40,000 lb., which seemed to be pretty near the maximum that could be attained within the 24-hour cycle, as it required about five hours to ladle, using as many men as could work to advantage.

"In 1895 an innovation was made, in the shape of large ladles, holding 200 lb. each and running upon a track — either overhead or on the surface of the ground. These ladles were dipped and managed by two men, and a further improvement was soon effected by which the copper could be tapped direct from the refining furnace into these large ladles. This method of casting

[1] Any discussion of the reason why it is necessary to leave something like three-quarters of a per cent of cuprous oxide in refined copper would be entirely foreign to the purpose of this work. The subject is obscure and still unsettled. I may say, however, that all experience and research, so far as I am aware, brings us to one, or both, of two conclusions; starting with positive knowledge that the cuprous oxide, *per se*, does not improve the quality of metallic copper, it is possible that reduction of this dissolved cuprous oxide below certain limits by the operation of poling, either (*a*) involves the reduction to the metallic state of minute proportions of foreign substances that still exist in the copper, and are comparatively harmless as oxides, but injurious when reduced to metals; or (*b*) permits the absorption by the copper of certain of the gases evolved in poling (namely, hydrogen and carbon monoxide) which have an injurious effect upon the metal.

There are certain facts known to me through personal experience which make it very difficult for me to accept the first of these two explanations, while the second theory also seems unsatisfactory, for reasons which would require too much space to elucidate.

A fuller discussion of this subject, and references to some of its most useful literature, will be found in "Principles of Copper Smelting," Chapter XII.

[2] These notes upon the Casting of Copper were kindly prepared for this book by Professor Arthur L. Walker, the inventor of the Walker machine for casting copper.

from the large ladles, while satisfactory for rough castings, did not answer for finer work, as the splashing of the metal caused 'cold sets' in the moulds.

"In 1897, however, a machine for casting fine copper was designed and installed in the electrolytic refinery of the Baltimore Copper Smelting and Rolling Co. by Arthur R. Walker, by which all shapes of refined copper could be cast successfully and rapidly, the molten copper being taken from the furnace on one side of the machine, and the cooled castings delivered by the conveyor at the other side. In designing the machine, the principal point kept in mind was to have the copper poured into the moulds in such a manner that there would be absolutely no splashing, and, to accomplish this result, a ladle was suspended under the tapping-spout of the refining furnace in such a manner that, in pouring from this ladle into the moulds, the fall of the stream would be reduced to a minimum, and would also be constant; or, in other words, that the mouth of the ladle should be directly over the moulds, as in the case of hand-ladling.

"Another feature consisted in bringing the moulds to the ladle instead of in carrying the ladle to the moulds. It was also considered necessary, at the outset, to allow the stream of metal to fall upon an intermediate deflector, which should distribute the molten copper in the mould, but this was soon discarded as superfluous.

"The casting apparatus consists essentially of a central horizontal wheel, or turn-table, revolving upon a set of rollers running on a circular track. From the rim of the wheel, arms are suspended which, in turn, support the moulds at their outer end, making a complete ring or circle, and adjustable, so that moulds of varying length can be accommodated. The moulds have trunnions on each end, which rest in recesses at the end of the arms, so that they can be turned over completely for the purpose of dumping the copper casting into the water-bosh.

"The ladle which receives the copper from the furnace and delivers it to the moulds is pivoted on trunnions placed as near as possible to the spout. These trunnions rest in curved supports fastened to a rocking shaft, so that by rotating this shaft the mouth of the ladle can be moved forward while the copper is being poured into the mould, and backward when rotating the wheel. The ladle is supported at the rear end by means of

a hook engaging in a yoke fastened to a flexible cable which, in turn, is connected with the piston of an hydraulic cylinder. By this means the ladle can be raised at the back for pouring, and lowered again to its normal position when the mould is filled to the proper point.

"In the center of the wheel is a stationary platform, around which the wheel revolves and upon which are placed the three levers that regulate the entire operation.

"Over the water-bosh into which the copper is discharged are two inclined planes. In rotating the machine, as the filled mould passes over the water-bosh, the dumping-arm attached to the rear side of the mould comes in contact with a finger on the first inclined plane, which turns the mould completely over, so that the copper casting drops out and into the bosh on the inclined plate arranged to receive it. On further rotating the wheel, the dumping-arm is caught by the second inclined plane, and the mould is returned to its original position.

"The copper casting which is dumped into the bosh slides down until it reaches the conveyor, which picks up the castings and delivers them at the head of the conveyor, discharging them onto a car.

"In the regular operation of the machine, a continuous stream of copper is allowed to flow from the furnace into the ladle. The operator in the center of the wheel rotates the machine by means of a controller governing the electric motor, until the mould arrives in position under the mouth of the ladle. In the meantime, by means of a second lever, the mouth of the ladle is brought over the mould, and the rear of the ladle is raised to pour the copper, this last operation being under the control of a third lever that operates a three-way valve. As soon as the mould is filled, the ladle is lowered to the horizontal position, moved back to clear the moulds, and the wheel rotated to bring the next mould into position. The operator becomes so expert that all three of these movements are taking place simultaneously, thus greatly increasing the rapidity of the process.

"In casting small ingots the mould must, of course, be stopped precisely at a certain point, and in this case, 8, 10, or 12 ingots are poured at once from as many spouts in the ladle, the spouts being so regulated that the flow of copper is equal from each. When casting long wire-bars, however, the copper is poured into

the moulds while the machine is moving. After the moulds are filled, the copper cools gradually, and the apparatus is designed in such a manner that, by the time the mould reaches the dumping point, the copper will be fully set and ready to be dropped into the water. The conveyor in the water-bosh is so arranged that, after the copper is dumped, sufficient time will elapse before it is discharged at the top, in order that the metal may be cool enough to handle.

"By a simple rearrangement of the arms, all of the principal shapes of refined or unrefined copper may be cast.

"The question of capacity is of extreme importance with the large charges now refined, and the following statement kindly furnished by the superintendent of a plant which is using six of these machines gives actual results:

"I have accordingly taken the actual work for six months, and find that it gives the following figures:

Anodes	65,000 lb. per hour
Ingot bars	51,000 lb. per hour
Ingots	50,000 lb. per hour
Wire bars	53,000 lb. per hour

"These figures include all delays in connection with difficulties in casting, so that the actual capacity of the wheel is somewhat greater. Our practice here has been to force the furnace, but give ample time for casting, in order to get good work, which is rather the reverse of the old-time idea. We allow, in general, from five to eight hours for the casting, the time varying with the character of the work."

The furnace-charge in the above case runs from 350,000 to 400,000 lb., and the maximum amount that can be cast per hour is about 75,000 pounds.

Fig. 104 shows the wheel arranged for casting ingots, eight being poured simultaneously from the multiple-spout ladle.

The operation of casting copper mechanically, as just described by Walker, presupposes a direct flow of the metal from the refining furnace, and the consequent necessity of having the stream of molten copper under absolute control during every moment of the process. Such control was impossible under the old method of tapping a furnace hearth from its deepest point, and then attempting to regulate the flow by plugging the tap-hole from

without. No one but a smelter can appreciate fully the extreme precariousness of attempting to execute any such manœuvre in the immediate vicinity of a stream of white-hot explosive metal, gushing from the tap-hole with an eighteen-inch head above it and several hundred thousand pounds back of it, ready to sweep everything before it if the operator has made a miss of his plugging operation. Moreover, the ball of clay used in plugging the hole soon bakes hard and becomes more or less saturated with malleable metallic copper, and the entire tap-hole, with its surroundings, soon gets into a perfectly hopeless condition.

This difficulty is overcome by drawing off the copper gently from the surface of the bath instead of allowing it to gush from its deepest point. This may be exemplified by comparing the situation to that of a pond of water, restrained by a dam, which it is necessary to empty completely, and yet to keep the outflowing stream under perfect control during every moment of its passage.

It would be a dangerous plan to drill a hole through the dam at its deepest point, and then attempt to control the stream as it gushed out with the full head of water above it. On the other hand — assuming the dam to be formed of suitable material — a safe and easy plan would be to dig a little ditch transversely across the dam down to the surface of the pond, and then permit a gentle stream to flow constantly, deepening the ditch gradually as the surface of the water lowered. A slight obstacle placed in the ditch would diminish the flow when desired, or a few shovels of clay would stop it completely and yet leave it in such shape that it could be started again at a moment's notice.

This is the simple principle upon which a great volume of molten copper upon the hearth of a refining furnace is kept under absolute control during its withdrawal. The precise details of the methods by which this principle is applied in practice are variable, and must be obvious to any person who is familiar with the behavior of melted copper. I append a brief description of the application of this principle at our large refineries.

As the drawing-off of the metal commences at the surface of the path, and as the gutter must be deepened gradually until the hearth is drained dry, it is plain that the tap-opening must consist of a vertical slot in one of the side-walls of the furnace, extending from the deepest portion of the hearth up to a point

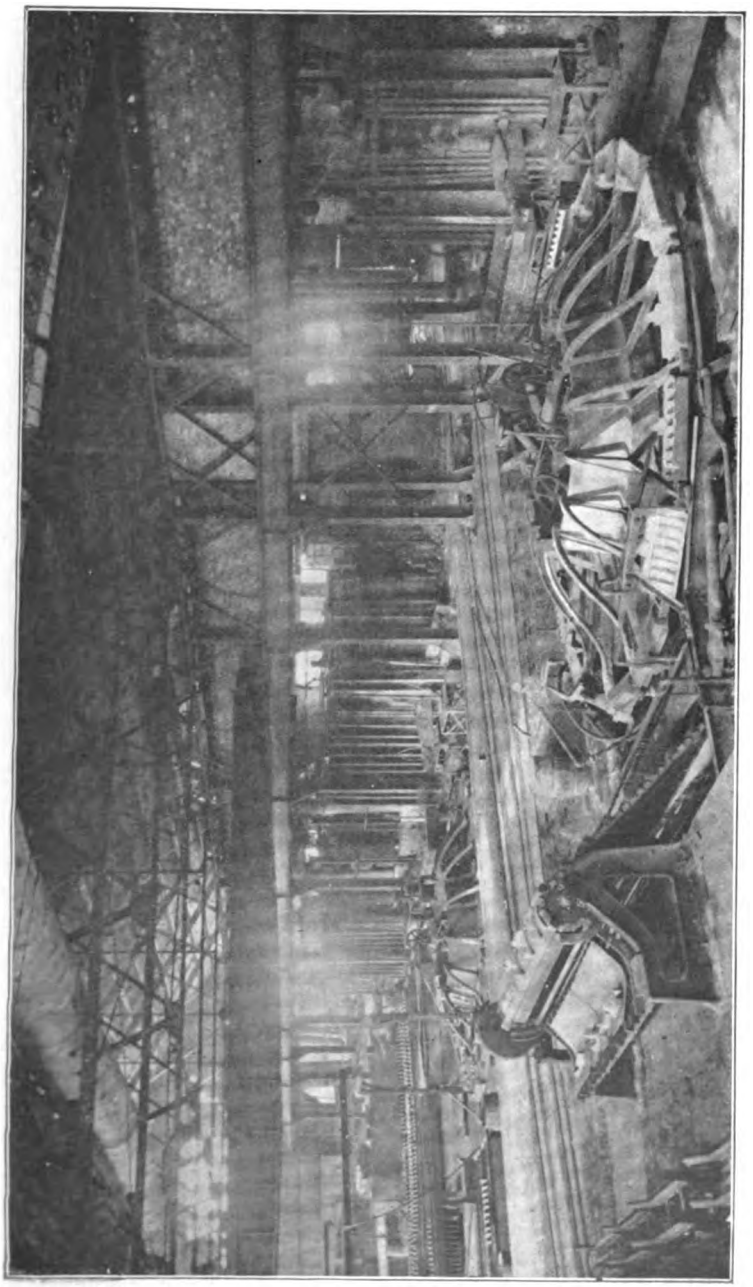


FIG. 104. — Walker casting machine — making ingots

above the highest possible horizon that the metal will ever attain. At the commencement of the refining, this tapping-slot must be filled with some substance that is capable of withstanding the weight and heat of the liquid metal, and yet can be scratched or cut away as the surface of the bath sinks. A finely-screened mixture of sand, clay, and coal is found suitable for this purpose and, being tamped in to the deep notch in the side of the furnace, is supported exteriorly by little transverse iron bars one inch square and about 16 inches long, spaced three inches apart, and simply laid in between lugs formed by one of the furnace-plates. When the time for casting arrives, the iron bars are removed and the clay mixture is cut down to the surface of the metal.

The flow of the stream of copper is checked by holding the end of a wooden pole in the gutter, or may be stopped entirely by introducing a dam in the shape of a piece of fire-brick fastened onto a split pipe.

The launders are of cast-iron lined with a mixture of cement, clay, and sand. The large tapping ladles are also of cast-iron, and have a three-inch lining of cement and sand coated with bone ash.

Ingots and wire-bars are made in copper moulds which are thrown back into the furnace as they become defective. Copper poured at too high a temperature will, of course, destroy these moulds rapidly. The cores around which the moulds themselves are cast are made of cast-iron and, in the case of the large wire-bar moulds, are water-cooled to prevent the cracking of the copper from the expansion of the core. The moulds are further protected by sprinkling them with bone ash.

As the slightest movement of the filled mould before the surface of the copper has solidified will cause a wave, and as this wave will chill on the side of the mould and produce a "cold set," it is customary, during the machine-casting of wire-bars, to chill the surface of the metal, before the wheel is started, with compressed air, admitted upon its surface through fine holes in pipes above the mould.

Having considered the six stages into which the refining process may be divided for descriptive purposes, I will go briefly over the operation as a whole, pointing out such matters as appear particularly useful.

A great deal of the rapidity and comfort of the refining process

in reverberatory furnaces, where bituminous coal is used as fuel, depends upon the care and judgment with which the firing is conducted. For instance: where coking coal is employed, a considerable saving in time and heat is effected by cleaning the grate and removing the clinkers in the following manner. During the two or three hours occupied in charging the hearth with copper, the fire-box is filled with fresh coal. Before the charging is completed, this coal has coked into an arched mass of sufficient stability to sustain its own weight. This permits the complete removal of the grate-bars, and the thorough cleansing of the fire without loss of heat, for the mass within the fire-box is in condition to produce a smelting temperature within a short time.

Before the oxidizing period, the fire-box is again filled full of coal, and the fire left undisturbed if possible, in order that reducing gases and sulphur may not be evolved by interfering with the fire in the middle of oxidizing.

When ready to pole, the grate is again filled up and left undisturbed, as any cutting or stirring of the fire tends to let in air and lengthen the time required for reduction. This is indeed the general rule in refining: that, so far as practicable, any interference with the fire should take place between, rather than during, the various periods.

The great capacity of the refining furnace of the present day — up to 500,000 lb. and more — is reached less by any phenomenal increase in size of furnace than by improvements in construction, by which the hearth is enabled to carry a much greater depth of melted metal than was feasible in the older furnaces, and is strengthened accordingly. In these latter, the bath of metal resembled rather a shallow lake with gradually shelving shores all around, while the large modern furnaces resemble rather a reservoir with perpendicular sides, the melted copper extending high against the enclosing brick walls. This is rendered possible by the use of a lining of magnesite brick — or similar material — and by the abolition of critical points of junction where the metal might work down and burst through the side-walls, or float up the bottom. This construction will be considered under the description of the furnace itself.

Many refining furnaces work only six days in the week, bank-

ing the fires over Sunday. In this practice, it is customary to charge the furnace with copper, as usual, on the Saturday night, put up the side-doors, and bank the fire; on Sunday afternoon firing is commenced, and by Monday morning the regular work is ready to proceed.

It will, of course, be noted that most of the great refineries execute two distinct sets of operations upon the copper which they receive; perhaps it would be more correct to say that they make two distinct products: anodes, for the electrolytic refining, and refined copper for the market (ingots, cakes, wire-bars, etc.). The copper of the anodes is not a finished product, as it must still undergo two operations: electrolytic refining, which produces cathodes of pure copper, and a final furnace-refining to make the metal ready for the market.

A tabulated form of this statement will make it more clear.

MATERIAL LIKELY TO BE RECEIVED AT A LARGE REFINERY, WHICH HAS ALSO AN ELECTROLYTIC REFINING PLANT

Material received		
Copper bars without gold or silver, and so pure that they do not require electrolytic refining.	To Refining Furnace yielding merchantable Copper.	
Cathodes (from outside electrolytic refineries).		
Copper bars containing gold or silver, or so impure that they require electrolytic refining. These are made into *anodes* at once.	Treated electrolytically.	Cathodes (to Refining Furnace).
Anodes (from outside sources).		Gold and silver slimes.

Many refineries purchase, or treat for a fixed price, gold- and silver-bearing metallic copper in the form of blister pigs, or converter-bars, or anodes, and it is a matter of the greatest importance both to buyer and seller to determine their exact content of precious metal. Variations in the value of the samples taken have been almost universally a cause of grave dissatisfaction and, without following the subject in detail, I will point out the

methods of sampling metallic copper that are now generally used, and the reasons for their adoption.

The late investigation of this matter at the Washoe smelter was conducted so carefully, and on so large a scale, that the points thus established carry an authority which makes it worth while to describe the results, as given by the superintendent, at some length.[1]

The anode plates produced at the Washoe smelter are made by pouring the converter copper direct into the refining furnace, where it undergoes a partial refining, and is brought to the proper pitch for casting into anodes. These plates are 36.75 in. long, 28 in. wide, and 2.06 in. thick. By reason of disagreement in the silver content of the anodes between the smelter at Anaconda and the refinery in New Jersey, it became necessary to investigate the methods of sampling employed.

At the smelter the copper is sampled by taking four samples, at intervals of one hour, from the stream of copper as it flows from the refining furnace to the anode moulds, each sample weighing four to six ounces. The sample is obtained in the form of shot-copper, by "batting" the stream of metal into water with a wooden paddle. The samples are dried, examined for particles of burnt wood, and screened on a 10-mesh screen of No. 8 wire to remove the fines, the oversize then being screened on a 4-mesh screen of No. 20 wire to remove the coarse. The undersize of this screening is taken for the sample. The four portions are then thoroughly mixed and reduced in the usual manner.

At the refinery, in New Jersey, the anodes are sampled by a method developed by Dr. Edward Keller, by which every fourth anode of a lot is drilled, using a 99-hole templet, as shown in the accompanying illustration. The holes in the templet are used in continuous order, one hole to each anode. For instance: if, in lot No. 890, holes from No. 1 to 90 were used, then in lot 891 the sampling would commence with hole No. 91. Before drilling, the anodes are carefully swept to remove foreign matter, and drilled with a half-inch (0.012 m.) drill completely through the anode, every portion of the drillings being saved. The drillings from the entire lot are run through a grinder until the sample will pass a 16-mesh screen. The sample is then thoroughly mixed

[1] "Sampling Copper Anodes at Anaconda," by William Wraith, Presented at Pittsburg Meeting of the Am. Inst. of Mng. Engineers, 1910.

and quartered, each quarter weighing about one pound. This is screened over a 40-mesh screen, and the undersize and oversize are weighed. The same ratio of coarse to fine is maintained in the portion weighed out for analysis.

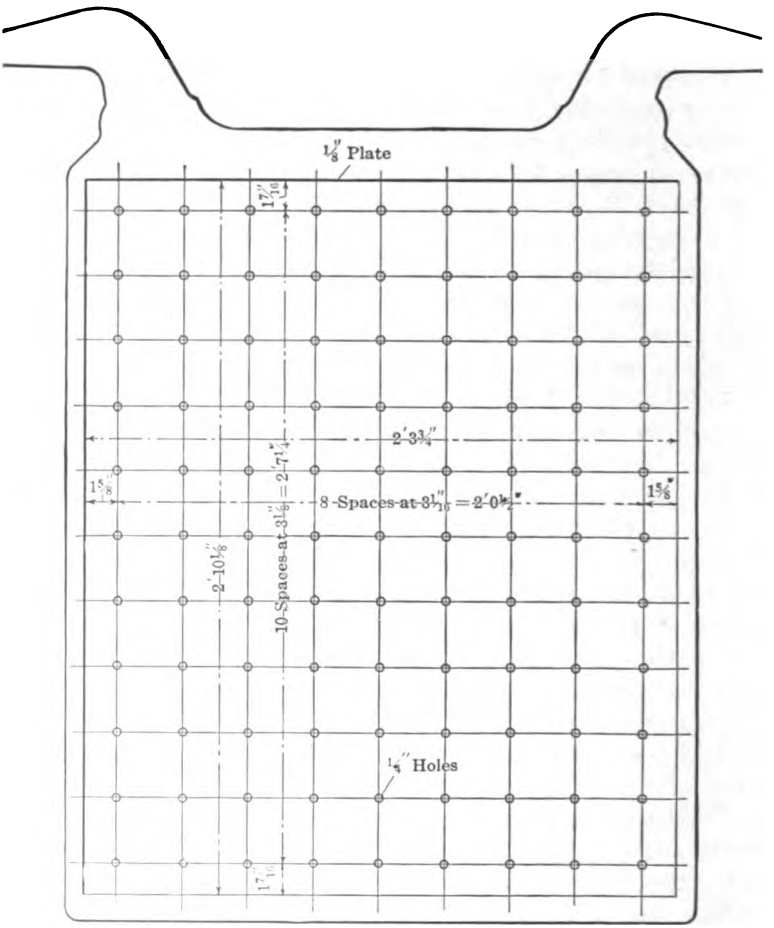


FIG. 106. — Templet for sampling anodes

A third method of sampling was also investigated at the smelter. This consisted in taking a ladleful of copper from the furnace, or furnace-stream, and shotting it by pouring it over a wooden paddle into water.

The correctness of the smelter sampling-method was tested

by taking shot samples of the pour each hour, the results for a typical lot being: 76.07, 75.95, 76.19, 76.37 oz. silver per ton. Testing the various portions of the flowing stream of another lot for possible segregation of silver gave: front of stream, 72.38 oz.; middle of stream, 72.29; back of stream, 72.28 oz. per ton.

Tests of the ladle-shotting method indicated that too high results would be obtained, owing to segregation of the silver. A charge of copper, of which the regular furnace-stream sample yielded 74.99 oz. silver per ton, showed from 76 to 81 oz. by the ladle-shot method, the chilled skull in the ladle assaying only 72.89 oz. per ton.

To test the uniformity of the anodes, six anodes were selected at equal intervals during the casting of a lot. The anodes varied from 71.21 to 71.68 oz. silver per ton. Two lots of anodes were taken from lot 911 for drilling samples. One of these was the regular drilling sample obtained by drilling one hole in every fourth anode according to the templet, beginning the drilling with the first anode cast. This showed a silver content of 74.31 per ton. The same lot was re-drilled, beginning the count with the third anode instead of the first. This showed a silver content of 74.5 oz. per ton. A comparison of the results obtained by the shot sample from the furnace stream and by the drilling method, on 28 lots of anodes of about 100 tons each, indicated that checking results could be obtained in this manner.

The conclusions drawn from these investigations were: (1) that the shot sample taken from the furnace stream and the drillings of every fourth anode, according to Dr. Keller's templet, will check within practical limits, and will give the true silver content of the copper; (2) that the ladle-shot method will give high silver results, due to a segregation of silver toward the portion that solidifies last.

In order to illustrate the routine of work as carried on at the smaller refining furnaces which are not provided with mechanical appliances for charging and casting, I will describe the practice at the refining department of the Boston and Montana smelter, at Great Falls, where cathodes from the electrolytic plant are refined into commercial shapes.

The inside hearth-measurements of the furnace are 14 ft. by 24 ft., the fire-box being six by seven feet.[1]

[1] One foot (12 inches) = 0.305 m.

The coal used is of poor quality, having, approximately, the following composition:

ash	22.9	per cent
volatile matter	29.3	" "
fixed carbon	46.5	" "
sulphur	1.0	" "
	99.7	" "

The furnace is built over a vault, and is provided with a sand bottom 28 in. thick at the bridge end, and smelted-in in two layers. It has a nine-inch slope toward the ladle-hole at the front end. There are checker-holes in the bridge for admission of air. The arch is constructed of 12-in. Anaconda silica brick.

At three in the afternoon the men begin charging cathodes through doors on both sides of the furnace. The cathodes weigh 110 lb. each, and two are charged on each paddle. The hearth holds about 80,000 lb. of copper, and the charge is melted and ready to skim at five the next morning. After skimming, vigorous oxidation is begun, compressed air at 13 lb. pressure being admitted through two one-inch pipes. The blowing takes little more than an hour to bring the charge to "set copper," and about two feet of iron pipe is used up on each side of the furnace during the process.

The bath is now covered with charcoal, and poling is begun. In about $1\frac{1}{4}$ hours the copper reaches the condition of "high blister," and a finishing poling of one hour usually brings it to the desired pitch. Where wire-bars are cast, tests are constantly made for electrical conductivity. This is effected by cutting a section out of a small test-bar, drawing it out to No. 12 wire (0.081 in.). These wires are annealed by passing a current of 130 amperes, and 40 volts, through the wire for one minute, which heats it to about 760 deg. C. The conductivity test is made with the Toopes Conductivity Bridge.

A torsion test is also made with the same size of wire, but in this test hard-drawn wire is employed. Two clamps are set six inches apart, and the wire is twisted until it breaks. This copper usually stands about 50 twists; the Bell Telephone Company's requirements are that it should bear 40 complete twists before breaking.

The tests for tensile strength and elongation are also made on

hard-drawn wire, the tensile strength of first-class coppers ranging from 65,000 to 66,000 lb. per square inch.

The casting is done with "bull-ladles" holding from 150 to 350 lb. copper, and running upon a trolley overhead, though guided and tipped by hand-power.

Metallurgists are occasionally asked if it is possible to use wood as a fuel in the refining of copper. This question may be answered positively in the affirmative, provided the calorific power of the wood is high enough to produce the required temperature. If the wood is capable of smelting an ordinary fusible ore, without too much waste of time, it is quite competent for melting copper, and its freedom from sulphur is a distinct advantage.

I have seen wood used, and have used it myself for this purpose, and the only serious difficulties were those arising from trying to use wood in a furnace designed for burning coal, and in which the fire-box was too small, and the volume of air passing through the grate too large. In Chapter XI in this volume will be found plans and directions for the adaptation of furnaces to the use of wood.

There is, however, some considerable difference in the way of managing the fire when using wood for refining. It is impossible to maintain the massive body of enduring coals in the fire-box that, in firing with coal, assists so greatly in maintaining a uniform heat and an unchanging atmosphere in the hearth. Even the most massive wood burns something like kindlings compared to coal, and it is impossible to avoid firing at frequent intervals. There is, however, none of the profound disturbing of the conditions of combustion that occurs when a coal fire is stirred and loosened, and men firing with wood learn to maintain their temperature and atmosphere fairly uniform, by always keeping a large fire in the fire-box and governing the heat and the reducing or oxidizing action by a damper at the top of the stack, or elsewhere.

An old conservative refiner who has been brought up with coal is prone to deny the possibility of refining copper properly with wood, and it is sometimes necessary to place the furnace in charge of an ambitious assistant. A charge or two of metal may be spoiled and require re-melting before the men learn how to

keep the copper up to pitch, as it is likely to fall back and lower its pitch until the peculiarities of the new fuel are grasped.

I need scarcely say that no satisfactory results can be anticipated where green or unsound wood is used. The calorific power of wood is none too high at the best, and it should be well-seasoned, with a fair proportion of massive sticks — split a single time to ensure drying — and with enough sticks of three and four inches diameter to fill the interstices and make a solid fire.

In Sweden, Russia, Siberia, and other countries where wood abounds and coal is expensive, regenerative gas plants, burning refuse wood, are used to heat reverberatory furnaces, and are found particularly advantageous for refining copper, both temperature and atmosphere being so thoroughly under the control of the operator. In those countries they are constructed in a simple and inexpensive manner, and are managed without difficulty by workmen whose intelligence certainly does not exceed that of our own operatives.[1]

While it is always advantageous and comfortable to use high-grade coal in heating a reverberatory furnace of any description, there is nothing peculiar about the refining of copper that demands better coal than other smelting processes, with a single exception: that is to say, the coal should carry as little sulphur as possible, else the absorption of SO_2 gas by the copper will continue as long as it is being evolved in the fire-box, and will disturb and prolong the sensitive "finish" of the operation.

At the Great Falls smelter, as I have already mentioned, a very poor coal with one per cent of sulphur and more than 20 per cent of ash is employed in producing first-class wire-bars from cathodes. The influence of the sulphur in the coal during the melting-down stage is greatly diminished by dipping the cathode sheets, before charging, in a bath of milk of lime.

It is also essential that the heating power of the coal shall be sufficient to maintain a reasonably uniform temperature for considerable intervals without frequent stirring or loosening of the fire, or removal of clinkers. The amount of fuel required is so

[1] In old "Modern Copper Smelting," 7th and later editions, I give a detailed description and plans of a little Swedish plant of this kind, with a description of its management in refining blister copper, as well as in smelting cement copper for blister. At this plant the fuel used consisted of old roots, ends of sawed timbers, sawdust, and other waste wood.

small in proportion to the value of the product that it is generally considered worth while to employ a good grade of coal in refining. This is only in harmony with the general rule that, the more concentrated and valuable the material treated, the more reasonable it is to spend money on it.

The amount of coal required to refine a given weight of copper varies with the calorific power of the fuel, the quality of the copper, the construction of the furnace, and the skill and energy of the attendants. The following figures represent the average of a considerable number of different works using fairly good bituminous coal; and indicate the advantage of large charges so far as the consumption of coal is concerned:

Weight of charge, lb.	Pounds copper refined by one pound coal
20,000	3.3
50,000	4.9
100,000	6.7
400,000	7.2

In the large anode furnaces, where high-grade converter bars are refined sufficiently to cast into smooth anodes, one pound of coal is sufficient for 7.75 lb. copper.

The construction of reverberatory furnaces has been studied in detail in the chapter on that subject, and there is little in the type of reverberatory furnace employed for refining copper to distinguish it from that used for smelting ore excepting the complete support given to the hearth by encasing it in iron plates. Such differences as there are proceed from difference in behavior of the material to be treated and might, indeed, be imagined in advance, to a considerable degree.

In the first place the two substances melt down in a totally different manner. Ore melts by a process of gradual liquation, the more fusible compounds melting quickly, while the others gradually soften as the heat rises, and continually readjust their molecular arrangement in harmony with the temperature prevailing at the moment until everything is liquid. Experience has taught that the heat of the fuel can be utilized best in a very long hearth, and that this long hearth also enables us to settle the matte globules out of the slag in a most satisfactory manner. The conditions in simply melting-down metallic copper are, of

course, radically different. It is an admirable conductor of heat; there is nothing to liquate, as the metal is already practically pure; there is no slag to form worth mentioning, and there is nothing to settle. Hence (if for no other reasons) refining furnaces have a comparatively short hearth. The width also is kept more moderate than with the ore furnaces, both for ease of manipulation in charging, and also because the great weight of the metal, as well as the depth of the bath that may be carried with safety, render great widths unnecessary.

Again, in ore smelting, the bottom, or hearth, is likely to be too cold, for reasons discussed fully in the chapter to which I have just referred. Consequently, the modern smelter — having learned how to construct his furnace on safer lines — has given up the cooling-vault, and builds his bottom with solid foundations. The refiner, however, dealing with 50 to 200 tons of heat-conducting, boiling metallic copper upon his hearth, finds that the latter is more likely to become too hot than too cold and, consequently, cools it by an underlying vault or, as in modern construction, sets the entire structure upon iron plates, resting upon a large number of small brick piers. This construction effects the double purpose of permitting the radiation of excessive heat from the bottom, and of detecting possible leaks or threatening burst-outs before they have advanced far enough to be serious.[1]

A few details from established plants will indicate how these principles find their application in modern practice.

The furnaces for refining cathode copper at one of the large Eastern refining plants have an inside hearth-measurement of 14 by 30 ft. The entire furnace rests upon masonry piers two feet square and two feet apart, there being thus a comparatively free space below the hearth some three feet in height. The accompanying cross-section of one of the melting furnaces at the Michigan smelter, Lake Superior, gives a general idea of this method of construction.[2]

The summit of each pier is capped by a steel plate, and upon these rest the $1\frac{1}{2}$-in. cast-iron plates that form the support of the

[1] There are still a few exceptions to the custom of cooling the bottom, some metallurgists letting it heat as much as it chooses, and relying upon great strength and thickness of construction.

[2] From a paper by L. S. Austin on "Recent Copper Smelting at Lake Superior," *Eng. and Mng. Journal*, Jan. 13, 1906.

furnace proper. Upon these plates the walls of the furnace are erected, and a flat inverted arch of fire-brick is constructed as a foundation for the sand bottom. The side-walls consist of 13½ in. of fire-brick, with six inches of red brick on the outside. Furnaces are heavily ironed with 11-inch I-beams, and, owing to the open space beneath the entire structure, there is no trouble with the lower tie-rods.

Where a sand bottom is used, as is generally the case with the Atlantic-coast refining furnaces, the inner lining is constructed of silica brick up nearly to the level of the lower surface of the

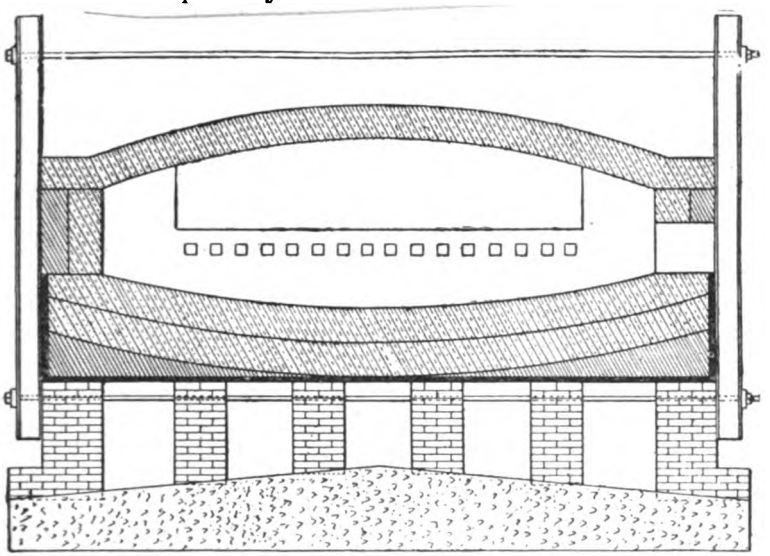


FIG. 107. — Cross-section of melting furnace at Michigan smelter

metal. From this point up to a short distance above the highest level of the bath, it is lined with magnesite brick, which resist the corrosion much better. The fire-box for a furnace of this size, burning good bituminous coal, might be eight feet square. The arch is of 12-in. silica brick and has a rise of 14 inches. The charging-doors are sometimes all on one side of the furnace; three in number, and about 36 in. in height and 64 in breadth. The conker-plate is thoroughly cooled, and the entire furnace is ironed and supported in the strongest manner.

The hearth or "bottom" is an important portion of the refining furnace, and has been the subject of much discussion and

experiment. Formerly the bottom was made universally of sand which, slightly agglomerated by an intense heat at the start, was allowed to absorb small quantities of copper gradually until a crust was formed of sufficient strength to withstand the operation of regular refining. This absorption of copper progresses slowly for an indefinite period, and the amount of metal that may be tied up in the bottom of an old refining furnace, with a deep sand hearth resting upon an unsatisfactory foundation, is something astonishing. I have been informed of one case where it was estimated that 424,000 lb. of copper was recovered upon tearing down an old furnace, although so large a quantity as this is unusual.

A certain limit is placed to this absorption by building the furnace upon piers, as just described; but, even under these improved conditions, a large and indefinite amount of copper is always tied up in the hearth.

Consequently, it is a growing custom to refine upon naked brick hearths, and I am inclined to think that, in time, this custom will become universal. I can count up over 20 refining furnaces with brick hearths with which I am personally familiar, and whose managers I have interrogated upon the subject, and no one of them has expressed the slightest desire to return to the sand hearth.

The following illustration, which the courtesy of the Anaconda Copper Mining Company enables me to offer, represents one of the two refining furnaces in use at their Washoe smelter, Montana, for partially refining the converter copper in order that it may make smooth and satisfactory anodes. There is still a third larger furnace in reserve, but the two of which this illustration is a type represent furnaces which are in constant use under about as severe duty as one is likely to encounter.

The bottom of this furnace is made of the local silica brick and, as it lasts at least four years under the usage that I shall presently describe, it shows the amount of mechanical wear that silica brick can withstand, when they are of good quality and laid properly.

The bottom is deeply concave, being 23½ in. deeper at the middle than at the sides. It has a total slope from fire-bridge to discharge-end of 1½ inches. It is composed of four inverted arches of silica brick set on end; the brick of the three inferior

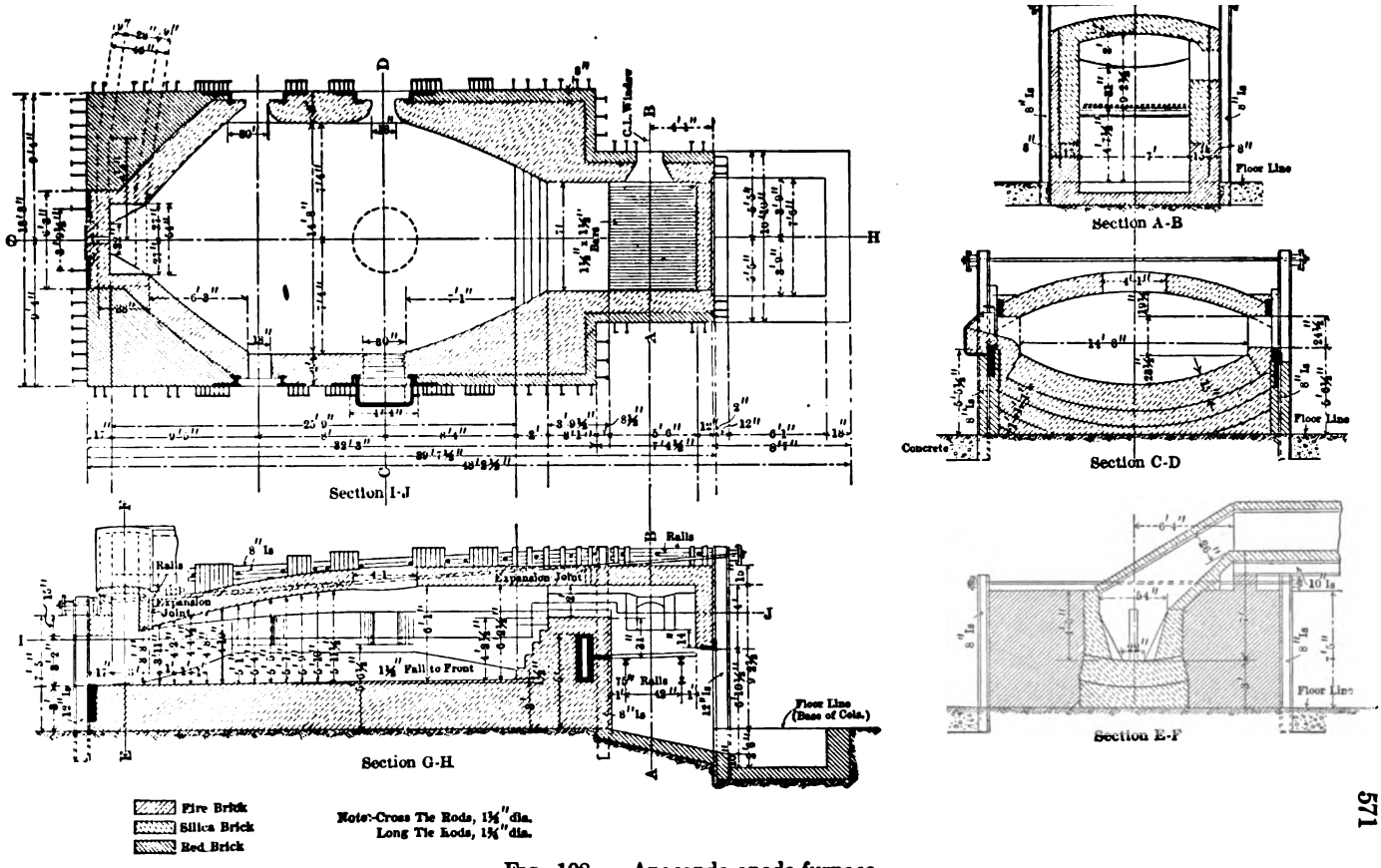


FIG. 108. — Anaconda anode furnace

inverts having a length of 12 inches, whilst the upper invert, with which the metal comes in direct contact, is 20 inches in length. The brick of these inverts are laid dry, and are swept over with finely-crushed Dillon quartz.

The fire-bridge is unusually massive and is tied strongly into the brickwork of the inverts as shown in the illustrations.

The arch is composed of 15-inch silica brick, and usually requires extensive repairs every four or five months, as the constant splash of the boiling metal fluxes it away steadily. I am informed that 20-inch silica brick are to be employed in the future. As already stated, the bottoms last at least four years, being repaired from time to time by throwing crushed quartz upon the empty hearth.

The duty imposed upon these anode casting furnaces is extremely severe and constant. They hold about 200,000 lb. of metal, and no sooner has the last anode of a charge been cast than the converter-ladles begin pouring in copper for a fresh charge. In order to complete their duty of 300,000 lb. copper (1½ charges per 24 hours), it is often necessary to begin blowing compressed air into the metal as soon as the bath is sufficiently deep to submerge the air-pipes, taking care, however, not to let the oxidation proceed too far before the entire charge is collected, else there is danger of such a violent evolution of SO_2 gas, from the reaction between cuprous oxide and the cuprous sulphide of the fresh converter copper, that there is danger of the melted metal being blown out of the furnace. Two of these furnaces are in constant use, with a third, and larger one, in reserve.

The converter metal contains about 98.3 per cent copper, and this process brings it up to about 99.3 per cent, at which stage it will cast into smooth anodes. The metal flows through the tapping-slit at the front end of the furnace into a suspended ladle, from which it is poured hydraulically into an anode mould attached to a platform conveyor operated hydraulically. When the mould is filled, the ladle is dropped to the horizontal position, and the conveyor is moved so as to bring the next mould into position. This apparatus casts 25 tons per hour. The copper is chilled by a spray and when "set" is dumped automatically from the mould onto a conveyor operating through a tank of water. The anodes are then ready for shipment to Eastern electrolytic refineries.

The preparation of a sand bottom for copper-refining furnaces differs but slightly from the similar operation in ore-smelting furnaces, except that even greater precautions are necessary at the commencement of its use, owing to the severe action upon it of the bath of boiling metallic copper.

The main requirement for bottom-sand is that it shall be fireproof and high in silica. I have already given various analyses of bottom-sands, and will add, merely, that even greater care must be taken in the selection of the sand for a refining-hearth. Experience has taught that certain local sands, or silicious rock, well known in the profession, are peculiarly satisfactory for this purpose, and it is better to bring a few carloads of absolutely satisfactory material from some distance than to add the risk of some cheap local sand to the multitudinous troubles that always accompany the starting of a new plant.

The hearth-material being ready, the first step is to calcine it thoroughly in order to remove all moisture, and to oxidize, and thus render inert, all foreign substances that may be present which are capable of oxidation. Otherwise, the deep-seated gases and vapors evolved when the bottom is first undergoing a high temperature might cause bubbles or porous regions in the mass, and the melted copper is certain to find and penetrate the weak spot, and perhaps float up great layers of the hearth. The sand is usually fired on while being gradually introduced and, when red-hot all through, and leveled properly, with a slight pitch from every point toward the tapping region, the actual melting-in of the bottom begins. The fusion-point of silica is but slightly above the temperature attainable in a well-designed reverberatory furnace, and 12 to 16 hours of continuous firing, aided by the slight fluxing action of the few per cent of bases in the sand itself, and by the ash of the fuel, forms a half-fused crust on the surface of the sand, varying from one-half inch to one inch in thickness. Unfortunately, the action of this excessive heat is not selective, and the arch, bridge, lining, and flue suffer seriously, which is one of the drawbacks of the sand bottom.

The fire is then slackened slightly in order to stiffen the crust, but no cold air should be admitted beyond what necessarily accompanies the succeeding manipulation. The bottom is now covered with a crust of sufficient stiffness to permit of the introduction of material intended to penetrate it and cement the loose

sand grains into a solid mass, like a monolithic block. This operation is called "seasoning" the bottom, and the seasoning material employed is usually of the same nature as the substance to be refined.

A one to two-inch layer of light scrap copper — usually from the electrolytic vats — is spread carefully over the bottom and fired upon gently. It will all be absorbed, and the hearth should then undergo another slight cooling, although not enough to allow it to contract and crack.

Then a somewhat larger charge of scrap is added, and the operation repeated. A certain amount of copper is oxidized and slagged by the silica, which aids in strengthening the bottom.

Under favorable conditions something like one-third of a normal charge is now introduced — no heavy blocks of copper being permitted — and is tapped out quickly, without refining as yet. The next charge may consist of some one-half or five-eighths of the normal amount, and is treated in the same manner. It should be weighed both in and out, and if there is no longer any considerable absorption of copper, a moderate charge may now be refined, getting it out as quickly as possible, and cooling down a little before the next one. It usually takes from three and one-half to five days to get a new bottom into good condition, and even then it should be treated tenderly for a week more.

As much of the cementing of the bottom-sand comes from the formation of cuprous silicate, the operation may be shortened by supplying a certain amount of the seasoning metal in an oxidized form. This is obtained from the copper rolling mills in the shape of black oxide, scale, etc. A ton of such material may be spread upon the freshly-leveled sand of a bottom that has been fired upon only six or eight hours, and then cooled slightly. This forms a more solid crust after a single melting than can be obtained by several treatments with metallic copper, but is only suitable for the temporary superficial crust.

The silica brick, which form the interior lining of the hearth-space up as high as the surface of the bottom, should in fact extend somewhat above it, as the magnesite lining which surrounds the metal bath will shrink away from the sand bottom, or from the windrow of fettling-sand. To prevent the fluxing action of the silica and basic brick upon each other at their line

of contact, it is advantageous to separate them by a layer or two of chrome brick, which is neutral to both.

As has been already explained, converter bars, blister copper, and all other varieties of metallic copper which are to undergo the process of electrolytic refining (either because they contain the precious metals, or because they contain too much arsenic, antimony, bismuth, selenium, or tellurium), have to be cast into the form of anodes of the standard size and weight suited to the vats in which the solution and electrolysis of the copper is to take place. Unless copper is tolerably pure it will not make smooth castings and, as much of the copper received at a refinery consists of bars direct from the converter process, it is necessary to give this a slight refining, during the melting operation for casting it into bars, mainly for the purpose of removing such sulphur as it still contains. It will be understood, therefore, that the object of this partial refining in the anode furnace is not to remove impurities from the metal except in so far as is necessary to obtain good anodes for the electrolytic work.[1]

The requirements of a good anode are:

1. Surface must be smooth, for several reasons:

(*a*) It is desirable to hang the anodes as closely together as possible in the tanks in order to lessen resistance.

(*b*) An uneven surface is likely to cause short-circuiting.

(*c*) If the surface is rough, slimes will accumulate on ledges, and slide off in minute avalanches, sticking occasionally to the cathode. Pure copper will plate over, and enclose, these slimes, and thus cause a loss in the precious metals, as well as favoring the extension of sprouts of copper extending from cathode to anode.

2. The plates should be of uniform weight, in order that all of the anodes in the vat may be used up at the same time.

3. The copper must be of good enough pitch so that the anode will support its own weight on the lugs.

[1] Converter copper that is sent to the refineries should not be cast in the form of deep pigs, with sloping sides. This is, practically, an obsolete method, as it renders sampling extremely difficult. The sampling-drills break in the long holes when they strike a lump of enclosed slag, whereas, in the shallow slabs which are now commonly used, such slag would float off during the casting. Converter plants which cast their metal in pigs, and then complain of disagreements in assay, should first correct their method of casting.

The labor required at the refining furnace varies according to the nature of the material, the size of the furnace, the arrangement of the plant, and the presence or absence of mechanical aids.

At one of the large Eastern refineries, where cathode plates are melted for wire-bars in a furnace having a capacity of 400,000 lb. and equipped with mechanical charging and casting apparatus, the ordinary furnace gang consists of:

one night furnace-man and helper
one day furnace-man and helper
two ladle operators
one mould painter
two fishers
four men to inspect and pile the finished bars.

There is also an operator for the charging machine, which has a capacity of 5000 lb. per minute when there are no delays.

The weight of slag from these large charges of cathode copper averages 2.7 per cent of the weight of the original charge. It contains about 40 per cent copper.

The weight of the slag from 220,000 lb. charges of converter slabs refined to anodes is 3.9 per cent of the weight of the original charge. This slag contains about 35 per cent copper and is smelted in the blast-furnace, producing blister copper of from 94 to 97 per cent copper, which goes back to the anode furnace.

The "Direct Method" of Copper Refining

It has long been known to chemists and metallurgists that when cuprous sulphide and cuprous, or cupric, oxide are fused together in certain proportions, they undergo double decomposition, yielding sulphurous acid gas and metallic copper. In other words, when raw white metal and calcined white metal are melted together, the only solid residue is metallic copper.

Messrs. T. D. Nicholls and Christopher James, of Swansea, have applied this reaction to the direct refining of copper matte, without any intervening operation, except the calcination of a portion of the matte, and the process has been in steady operation for some 20 years in the works of the Cape Copper Company, Ltd., at Briton Ferry, South Wales.

The remarkable improvements in the converter process, and

the establishment of the fact that silver losses are not high even when blowing highly argentiferous matte to blister copper, have prevented the introduction of the "direct process" into the United States; hence I do not feel justified in according it such space as its merits deserve. It is described at length in "Modern Copper Smelting."

CHAPTER XVI

FLUE-DUST AND SMOKE

THE gases evolved from the ordinary furnace operations of a copper-smelting plant are important for two reasons. In the first place they contain valuable substances and, secondly, they contain compounds which may be injurious to vegetable (rarely animal) life. Before attempting to study them in detail, it is essential to learn of what they may consist.

It is plain that they can contain no elements excepting such as already existed in the ores and fuels used in the furnaces or in the atmospheric air which forms the greater portion of their volume. These elements, however, have been rearranged to a considerable extent by heat and other influences, and the smoke thus contains many compounds which did not exist as such in the materials from which it was derived.

Considering them briefly, we may first dispose of the more simple and obvious constituents, such as water vapor, unchanged air, residual nitrogen from the air whose oxygen has been used up in the various processes, carbon monoxide, carbon dioxide, and other products of combustion, and — finally — the two common volatile compounds of sulphur and oxygen — sulphur dioxide and sulphur trioxide.

Furthermore, we shall find such particles of the ore itself as are light enough to be carried along by the draft, as well as many even lighter particles resulting from the formation of new compounds — this entire class of material being termed "flue-dust."

Finally, we shall discover a certain amount of material known technically as "fume," and consisting of solid matter in an almost infinitely fine state of division, which has resulted mainly from chemical reactions within the furnaces. Its composition, naturally, depends upon the ores from which it was derived, but consists usually of sulphates — and other salts — of lead, zinc, copper, iron, etc., with arsenates, arsenious oxide, and arsenious sulphide when arsenic is present in the ore. It generally contains

silver, and sometimes a little gold. Two characteristics of this peculiar product that are of great importance in practice are that it cannot be satisfactorily settled from the gases by a mere increase in flue area, and that it will not mix with water by mere contact. A drop of water falling through a fume-laden body of gases will simply push aside the particles of fume without wetting them, and without changing its own form.

There remains still the strictly gaseous portion of the smelter-smoke, and the make-up of the entire gas-current is now sufficiently elucidated to permit of an obvious classification for purposes of convenience: 1. Flue-dust; 2. Fume; 3. Gases.

1. *Flue-dust.* — As this material consists of fine solid particles suspended in the current of gases which is passing from the various furnaces toward the stack, and as these particles must be heavier than the gas-current that is supporting them, it would appear that they might all be settled completely by checking the flow of the current sufficiently to allow their specific gravity to assert itself fully — and this assumption is correct.

Unfortunately, however, the area required to reduce the velocity of the gas current to such an extent that the very finest particles of dust shall be deposited is so large that, in view of the great capacity of modern plants, this plan alone would be impracticable. It is, therefore, often aided by devices which assist in settling the dust particles, and which in some cases are equally important in promoting the precipitation of a portion of the "fume"; so that the study of the behavior of the finer portion of the flue-dust includes also a consideration of the more tangible portion of the fume.

A mere increase in area of the furnace-flue — usually effected by the introduction into the flue-system of a dust-chamber having a considerable length and a much greater cross-section than the flue itself — lessens the velocity of the gas current to such an extent that a large amount of its suspended particles can fall at once to the floor. This material represents the coarsest and heaviest grains of the flue-dust — that portion that even the most crude and imperfect plant is expected to save.

Some idea of the relative size of the flue-dust particles, as their point of deposition recedes from the furnace, may be gathered from determinations made in the dust-chambers at Cananea, Mexico, where blast-furnaces smelting some 50 per cent of fines

— material less than $\frac{1}{4}$ in. (0.006 m.) in diameter — are run with light wind pressure.[1]

Sample No. 1 of the following table was taken from the first main flue into which discharge the goose-necks from the eight blast-furnaces. This flue has a length of 280 ft. and a cross-section of 89 sq. ft., and contained 50.3 per cent[2] of the total dust deposited in the entire recovery chambers. A sample of it classified on screens having, respectively, 10, 20, 40, 60, 100, 150, and 200 meshes per linear inch, yielded the results shown under sample one of the following table. The remaining nine samples were taken at intervals in a main dust-chamber which succeeds the main flue that yielded No. 1 sample, and mark the diminishing size of the particles as the distance from their source increases, until the last sample — No. 10 — nearly all passes through a 100-mesh screen, and contains 38 per cent of material so fine as to pass a 200-mesh screen.

TABLE SHOWING PER CENT OF FLUE-DUST LEFT ON THE VARIOUS SIZES OF SCREENS

Number Sample	Per Cent on 10	Per Cent on 20	Per Cent on 40	Per Cent on 60	Per Cent on 100	Per Cent on 150	Per Cent on 200	Through 200
1	0.8	11.2	16.2	36.6	21.6	9.0	2.9	1.7
2	0.2	2.2	8.2	37 4	31.6	13.5	5.0	2.4
3	nothing	0.7	2.4	38.7	35.8	11.2	7.3	3.9
4	—	0.7	3.2	42.7	25.6	15.9	7.2	4.7
5	—	0.2	0.6	16.1	39.0	27.2	9.0	7.9
6	—	0.1	0.2	15.3	35.1	29.0	11.7	8.6
7	—	nothing	nothing	4.0	19.9	38.4	18.5	19.2
8	—	—	—	3.0	18.4	37.9	19.4	21.3
9	—	—	—	1.6	16.8	36.0	20.4	25.2
10	—	—	—	0.3	5.0	29.0	27.6	38.1

While the mere expansion of the flue into a large dust-chamber will cause the deposition of almost the whole of the suspended particles of flue-dust, provided the velocity of the current is reduced excessively — say to a speed of four feet per second —

[1] These figures are taken from a paper by Charles F. Shelby, Supt. Reduction Division, Cananea Consolidated Copper Company, in the *Engineering and Mining Journal* of Jan. 25, 1908, entitled "The Deposition of Flue-Dust."

[2] One foot (12 inches) = 0.3048 m.

the cost and cumbersomeness of such a system becomes prohibitory at our large modern smelters where one to two million cu. ft. of gases per minute are discharged through the central stack.

This has led to the introduction of various obstacles into the flues and chambers to aid in the settling of the dust, their effect being due largely to the eddies thus set up and to skin-friction. These obstacles, however, cause interference with the draft and cannot be pushed to an extreme, but they seem to be absolutely necessary when handling very large volumes of gases, in order that the area of the settling spaces may be kept within reasonable limits.

As it is difficult to discriminate between the finer grains of actual flue-dust and the coarser particles of fume which condense from gaseous into solid condition as the current cools in its passage through flues and chambers, it is particularly instructive to examine a case where the comparative absence of Pb, Zn, As, and Sb guarantee an almost complete immunity from fume, and afford an unobscured view of the behavior of the flue-dust proper.

Such an instance is afforded by the extended series of experiments upon the settling of flue-dust recently completed at the Copper Queen smelter, Arizona. I regret that I am obliged to compress those authoritative, and permanently valuable, results into so inadequate a space.[1]

These experiments were conducted on a large scale at the Copper Queen smelter with the object of determining whether it might not be possible to settle very fine dust from a current of gases by reducing the velocity of the flow. For this purpose, a small flue was built — 100 ft. long, with a cross-section of 23 sq. ft.[2] through which gases were drawn by means of an exhaust-fan, and then discharged through flannel bags to collect such material as was not settled previously. The gases were passed through this chamber at various speeds, ranging from less than one foot per second to over five feet per second. All of the dust was fine, approximately 90 per cent of it passing through a screen with 200 meshes to the linear inch.

[1] The brief notes upon these experiments here introduced are taken from a paper by George B. Lee, superintendent Copper Queen Smelting Works, Douglas, Arizona, in the *Engineering and Mining Journal* of Sept. 10, 1910, p. 504.

[2] One foot (12 inches) = 0.3048 m.

In the early experiments it was found that ordinary woolbags would not arrest the finest dust, and microscopic examination showed that a large proportion of the spaces between the meshes of the cloth were larger than the particles of dust which it was desired to recover. The closest-woven woolen material was then used, and the calculations of results were based upon the supposition that the entire amount of dust in the gas-current was collected either in the flue or in the bags.

I append the result of a single one of the many experiments reported in this paper, with the object of showing the decrease in quantity, and corresponding increase in fineness, of the dust particles deposited at increasing distances from their source. The silver and gold are given in ounces per short ton. (One ounce per ton (0.0034286 per cent).)

DUST CHARACTERISTICS — EXPERIMENT XVI — 4

	Amount of Dust Deposited		Per Cent Retained on Screen of			Per Cent Passing 200 Mesh Screen	Composition			
	Pounds	Per cent	100 Mesh	150 Mesh	200 Mesh		Gold Ounces	Silver Ounces	Copper Per Cent	Lead Per Cent
Division I	75.31	20.19	2.12	14.62	13.25	70.01	0.025	2.42	10.85	0.9
Division II	92.63	24.83	1.12	8.40	6.25	84.23	0.025	2.52	10.90	0.9
Division III	68.69	18.41	0.13	0.75	2.00	97.12	0.03	2.72	11.10	1.0
Division IV	26.12	7.00	0.25	0.62	0.50	98.63	0.02	2.98	10.95	1.7
Division V	13.88	3.72	0.25	0.25	0.13	99.37	0.04	3.16	10.70	2.8
Division VI	8.31	2.23	0.25	0.37	0.13	99.25	0.03	3.17	10.40	3.3
Division VII	5.50	1.47	0.37	0.38	0.25	99.00	0.025	3.27	9.70	3.6
Division VIII	4.56	1.22	—	—	—	100.00	0.025	3.37	9.40	4.5
Division IX	4.63	1.23	0.13	0.13	0.12	99.62	0.025	3.37	8.75	5.3
Division X	3.56	0.97	0.13	0.25	0.12	99.50	0.02	3.58	8.00	7.0
Total box	303.19	81.27	—	—	—	86.86	0.026	2.69	10.80	1.39
Bags	69.88	18.73	—	—	—	—	0.02	3.93	4.95	15.9
Grand total	373.07	100.00	—	—	—	—	0.025	2.92	9.70	4.1

The conclusions reached after long and exhaustive experimenting are so important that I quote Mr. Lee's words: "We believe, as a result of these experiments, that flues or chambers to settle dust need not be long. In fact, 125 ft. would appear to be enough if the velocity of gases through them does not exceed 150 ft. per minute, and that this speed may be materially increased if wires or screens are placed across the direction of the flow.

"These results are for the Copper Queen conditions, and are

not supposed to cover conditions where there is considerable lead and zinc on the charge. Under such conditions, undoubtedly, a larger proportion will reach the bags. The valuable contents of our ores are mainly copper, with a little gold and silver.

"If a chamber of large area, properly designed as to inlet and outlet, will settle the fine dust, it then becomes a question of how far it will pay to go in this direction. It has the advantage over the bag-house, or any mechanical appliance, that there is no machinery to operate and maintain. A chamber is being constructed at the Copper Queen works to handle the gases from two furnaces on the lines indicated, and it is the intention to test this question on a working scale."

It must be understood that, while the marked efficiency of a mere naked enlargement of the flue area — in other words, a dust-chamber — is indisputable, its installation and management must be accompanied by judgment and careful watching, and that the manner of ingress and egress of the gases, as well as their behavior within the chamber, must be considered with the greatest care, and regulated until the best possible results are obtained. Baffle-plates close to the inlet are nearly always essential in order to check the entering current promptly, and before it travels so far as to render useless the corresponding portion of the expensive chamber.[1]

[1] Reliable comparative results from actual practice are so valuable that I introduce a statement by Mr. Thomas Kiddie, late superintendent of the Tyee smelter, British Columbia; *Trans. A. I. M. E.*, Vol. XL, p. 900:

"With reference to the subject of the deposition of flue-dust 'by decreased velocity,' the members of the Institute who visited the Tyee smelter in 1905 were shown an experiment there being carried out on the lines of decreased velocity — first, for the purpose of lessening the amount of cold air drawn into the dust-chamber, and second, for the better recovery of the flue-dust.

"The draft was regulated by placing a damper in the stack and turning it down until the furnace-gases came out of the doors of the furnace; it was then raised sufficiently to remove all the gases at the lowest practicable velocity. The dimensions of the dust-chamber were 155 by 10 by 8 feet. A comparison of the recovery of flue-dust for two periods each of two years shows:

"First period, with a velocity of 1225 ft. per minute, gave 2.007 per cent flue-dust. Second period, with a velocity of 440 ft. per minute, gave 3.102 per cent of flue-dust.

"These results show that with decreased velocity there was an additional

Having learned that by far the greater proportion of the genuine dust particles may be settled, if it is practicable to reduce their velocity to four feet per second — or perhaps somewhat slower still for difficult cases — and that, consequently, large cross-sectional area rather than length is the important feature in the construction of chambers for this purpose alone, we come to the point where it will be necessary to begin to reckon with the "fume" that is apt to play such an important part in our smelter losses, as well as in the injury effected by our smelter smoke.[1]

2. *Fume.* — As I mentioned at the beginning of this chapter, the substance known to the metallurgist as "fume" is derived from the sublimation of material occurring in the furnace-charge, and consists of solid matter in an infinitely fine state of division. Its composition corresponds to the substances from which it was derived, and it contains, usually, sulphates, and other salts, of

recovery of 54.50 per cent. The value of the flue-dust was $14.92 per ton.

"A second plant has been similarly altered with equally satisfactory results. In this case there were smelted copper-gold ores in which the gold-values were much higher than at the Tyee smelter. Moreover, the size of the dust-chamber was much greater: namely, 300 by 16 by 10 feet. A comparison of the recovery for three years at high velocity with that of one year under decreased velocity gives:

"First period (three years), with a velocity of 1250 ft. per minute, gave 2.91 per cent of flue-dust.

"Second period (one year), with a velocity of 450 ft. per minute, gave 4.39 per cent of flue-dust.

"The additional recovery under decreased velocity was, in this case, 50.8 per cent. During the latter period, the higher saving in flue-dust was verified by the decrease in metal-losses. The value of the flue-dust was $30.20 per ton. In this second case the flue-opening from the dust-chamber into the stack, situated midway between the bottom and the top of the chamber, was reduced in size from 16 by 10 ft. to 2.5 by 10 feet."

[1] The treatment of flue-dust is discussed in the chapter on "Fines." From its nature and source it is plain that it is very bad practice to feed it back in dry condition into the blast-furnace. If there is not a reverberatory available for its treatment, it should be agglomerated with converter slag, or sintered in pots with sulphide fines, or briquetted with some binder — preferably concentrator-slimes. If none of these methods are practicable, it should be moistened thoroughly with water, and fed while still wet to the blast-furnace. If acid waters are available, it may be thoroughly wetted with these and, under favorable conditions, will often form sulphates enough in a few days to cement together in a very advantageous manner.

lead, zinc, copper, iron, etc., with arsenious oxide and sulphide when the ores are arsenical. Silver is usually present and, occasionally, a little gold. Parts of it do not condense from their gaseous condition into solid vaporous form until the temperature of the current of gases has been lowered materially. This is one of the most difficult classes of material with which the smelter has to deal, as it cannot be settled by any practicable slowing of the gas-current — like the flue-dust proper; nor can it be diluted with atmospheric air to such an extent as to become comparatively harmless — like the sulphur gases.

In order to deal effectually with fume — or with the gases that will form fume when they are cooled sufficiently to permit its condensation — it is necessary, first, to cool the gases so that the fume can form, and next, to catch it after it has formed.

The methods for saving fume which are of sufficient importance to demand consideration in a book of this nature are:[1]

1. Settling the fume from the gases in flues or chambers by retarding the velocity of the current.

2. Cooling the gases by one or more of several methods: radiation, admission of air, spraying with water, or passing the gases through centrifugal washers.

3. Subjecting the gases to filtration or friction (such as bag-filters, baffles of metal sheets, or wires hung in the flue).

4. Precipitating the fume by passing the gases through a space charged at numerous points with static electricity.

Referring to (1), it has already been stated that a mere retardation of the velocity of the gas-current, although reasonably (if not absolutely) effective for the deposition of the flue-dust particles, is not sufficient for the removal of the fume. The presence of extensive frictional, or eddy-forming, surfaces — in connection with cooling — is required to obtain any kind of satisfactory result, and the method then soon resolves itself into (3).

In the cooling system (2) the result is obtained either by wet methods or dry:

(*a*) Wet methods of cooling. Water-spraying in showers, or

[1] F. T. Harvard in *Bulletin* 44, for August, 1910, of the Am. Inst. of Mng. Engrs., publishes a paper entitled "The Condensation of Fume and the Neutralization of Furnace Gases" which contains so good an enumeration of methods that I make free use of it in the following discussion of the subject.

in besprinkled Glover towers; forcing the gases through a bath of water; passing the gases through centrifugal washers.

(*b*) Dry methods of cooling. Cooling the flues by circulating water about them; by presenting cool obstacles to the passage of the gases, such as beds of coke, chequer-work of bricks, metal wires, or plates, suspended in the gas-current; by the introduction of cold air to the flue or dust-chambers; and, finally, by building flues of great length, and with thin walls, to secure cooling by radiation.

The wet methods (*a*), in anything like their present form, are absolutely unsuited to the conditions obtaining at most of the large smelters in North America. The Theisen scrubber, so largely used by iron smelters in spite of the considerable power required to drive it, derives its main value from cleansing gases for power, accomplishing this result so thoroughly that they are used direct in gas-engines when desired. This would appeal to no copper plant, in America, with which I am familiar, and is applicable at Mansfeld only on account of the peculiar ore-charge there smelted, which yields gases almost suitable for fuel and makes it worth while to add a little superfluous coke to bring them up to the required standard.

The dry methods (*b*) of cooling the flues for the deposition of fume comprise a considerable list of devices, many of which are too elaborate and expensive for ordinary conditions and, especially, for large volumes of gases. In American practice they are reduced mostly to cooling by radiation, or cooling by the admission of air into the flues, or by a combination of both.

The presence of arsenic in the gases adds seriously to the difficulties of the metallurgist, as it is almost impossible to condense it effectually by any ordinary system of cooling, although such a large proportion of it may be removed that the injurious effects of the escaping gases are greatly diminished. In default of some radical cooling method — such as freezing by Gayley's method — the complete removal of arsenic from the fumes of large smelting plants seems impracticable, in the present state of the art, except by passing the gases through bags before permitting them to escape; this, however, would involve an almost prohibitory expense for a large plant.[1]

[1] Estimates made at Anaconda, Montana, of the cost of adding a bag-house to the Washoe smelter, of sufficient capacity to filter its escaping gases,

Despite the fact that arsenic cannot be removed completely by any ordinary reduction in temperature of the gases, the moderate cooling effected by measures which are entirely practicable is of great importance in the condensation and deposition of other fume-constituents, as well as in the recovery of the finer particles of dust which failed to settle nearer their source. The shrinkage in volume of the gases is, of course, an important element in the operation.

At some plants this cooling by radiation is effected by conducting the gases through long flues having a very thin roof and walls, in some instances only of the thickness of one-half a brick, and strongly buttressed. In other cases, after cooling the gases in brick flues down to at least 350 deg. C., they are conducted through long flues of boiler-iron where the radiation is great. On the whole, however, the use of baffled and air-cooled flues and chambers seems the most economical and satisfactory, and if to this can be added a sufficient bag-house, the recovery of metal values, as well as the removal of all injurious substances excepting SO_2 gas, may be regarded as complete.

A brief survey of a few modern installations in this country will illustrate the application of the principles enumerated in the preceding pages.

One of the most interesting of these installations is the recently-constructed system of the Boston and Montana Company's copper smelter at Great Falls, Montana. This plant treats some 4000 tons of ore daily, and contains blast- and reverberatory-smelting furnaces, roasters, converters, refining furnaces, etc. Its former stack- and flue-capacity being insufficient for the increasing demands upon it, which amounted at times to 1,500,000 cu. ft. gases (at 315 deg. C.), a new system was constructed which should provide for large future increase by being designed for a capacity of 4,000,000 cu. ft. gases per minute, at a velocity through the main dust-chamber of not more than 500 ft. per minute, and that the precipitation of the dust and fume should be facilitated by the use of Roesing's wires.[1]

showed that it would amount to $2,750,000, with an operating cost of $1850 per day. The additional arsenic recovered therein would have a value of about $204 per day. E. P. Mathewson's testimony before Judge Hunt, of the U. S. Court, 1909.

[1] By the permission of *Mines and Minerals* I make use of the accompanying cuts, as well as of certain information, from a paper by R. L. Herrick, in the issue of the above journal for December, 1909, page 257.

The chimney is interesting from its size and general construction. It stands upon an eminence some 245 ft. above the smelter,

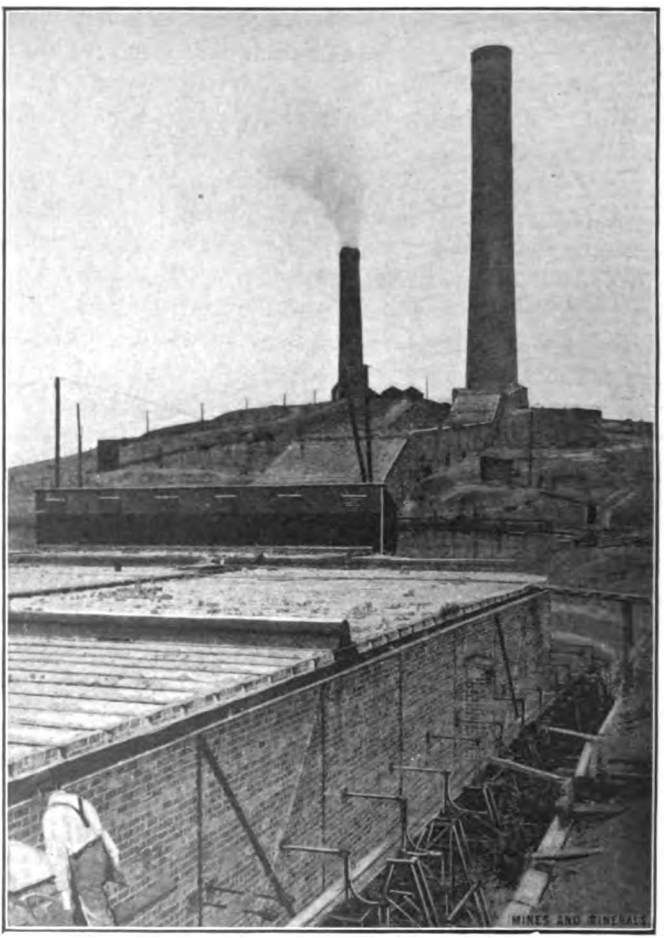


FIG. 109. — Old and new chimneys at the Great Falls smelter

and has an additional height of 506 ft., with an inside diameter of 50 feet.[1]

It is shown in Fig. 109, where also may be seen a portion of the main dust-chamber with the shaking mechanism for the wires.

The base of the stack is formed of slag-concrete mixed in the

[1] One foot (12 inches) = 0.3048 m.

proportion 1:3:5, and required 5200 bbl. cement, 2000 cu. yd. sand, and 4000 cu. yd. crushed slag. The chimney proper required 13,000 tons of radial brick, 3075 bbl. Portland cement, 5225 bbl. of lime, and 4180 cu. yd. of sand. The special brick were made upon the spot, and their features are shown in Fig. 110.

Upon the completion of the chimney proper, it was lined with acid-proof blocks laid in a special mortar containing sodium silicate, asbestos wool, etc.

It has an effective draft equivalent to $3\frac{3}{4}$ in. water, and an approximate capacity of 150,000 boiler horse-power, if it were to

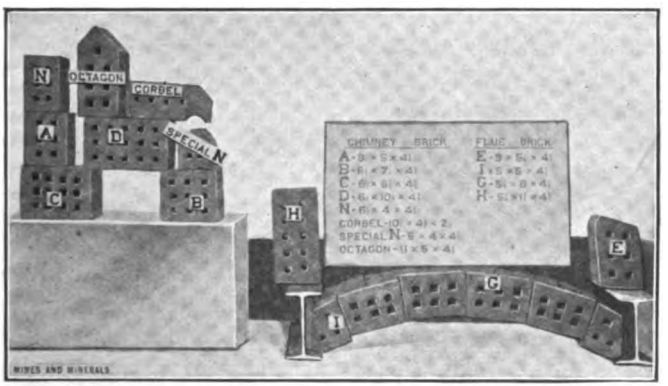


FIG. 110. — Special brick used in chimney

be used for boiler-plant operation, burning five pounds coal per boiler horse-power.

Referring to Fig. 111, it will be seen that the gases from most of the departments of the smelter are collected in a main cross-flue (*a*), and conducted above the buildings to the large dust-chamber (*b*). This is connected with the stack by the carrying-flue (*c*), which is 1200 ft. long in horizontal projection, and is 45 ft. wide and 21 ft. high inside.

The flues, as well as the dust-chamber, are framed of structural steel, built in by walls of brick. The roof is formed by jack-arches sprung between I-beams, the lower surface of the steel remaining uncovered, as it is not corroded by dry furnace gases. Expansion joints are provided at intervals of about 90 ft. (see Fig. 109), and the entire roof is rendered waterproof by cement. As it is intended that suspended particles shall be

deposited before the gases reach the long connecting flue, the latter is not equipped with collecting-hoppers for dust.

The great dust-settling chamber has an extreme length of 478 ft., and a width of 176 ft. over most of this distance. It is 21 ft. high inside above the collecting hoppers, and has below these a basement 12 ft. in height (Fig. 112), into which the 1040 hoppers, centrally placed above tracks, discharge their dust into trains of cars which convey it to the level above the reverberatory smelting furnaces. The cross-take (*a*) is provided with 100 similar dust-hoppers, which are served by a traveling crane.

The settling chamber is divided by a longitudinal brick partition into two equal parts, thus permitting the independent operation of one side while the other is being cleaned or repaired. At present, the average temperature of the gases at their entrance is 260 deg. C. and their velocity is 300 ft. per minute.

Two separated areas in this large chamber are hung with Roesing wires, suspended from a steel wire net bolted to the I-beams of the roof, and having apertures $1\frac{5}{8}$ inches square. The suspended wires are about 20 ft. long and are spaced $2\frac{1}{4}$ inches apart. They are of Nos. 10 and 12 Birmingham gauge, and weigh nearly one pound each, and as there are about 1,219,000 wires, the total weight of the suspended metal is some 608 tons.

The first area filled with wires begins close at the entrance of the chamber, and extends for 150 feet; then comes a gap of 50 feet, and then 150 ft. more of wires. In this vacant space, air-ducts from roof and basement are provided of such capacity that the temperature of the gases is reduced below the condensation point of arsenic and other vaporized substances. The purpose of this arrangement is, of course, to collect the mechanically suspended dust in the first portion of the chamber, leaving the condensable substances to be recovered in the succeeding area. At the inlet and outlet of the dust-chamber, as well as at the base of the stack, there are recording electric pyrometers and vacuum gauges. The entire cost of the system here described is said to have been about $1,100,000.

The selection and designing of the plant was preceded by elaborate experiments, upon an extended scale, and the use of wires was found, on the whole, preferable to baffle-plates, Freudenberg plates, expansions and contractions of passages, etc.

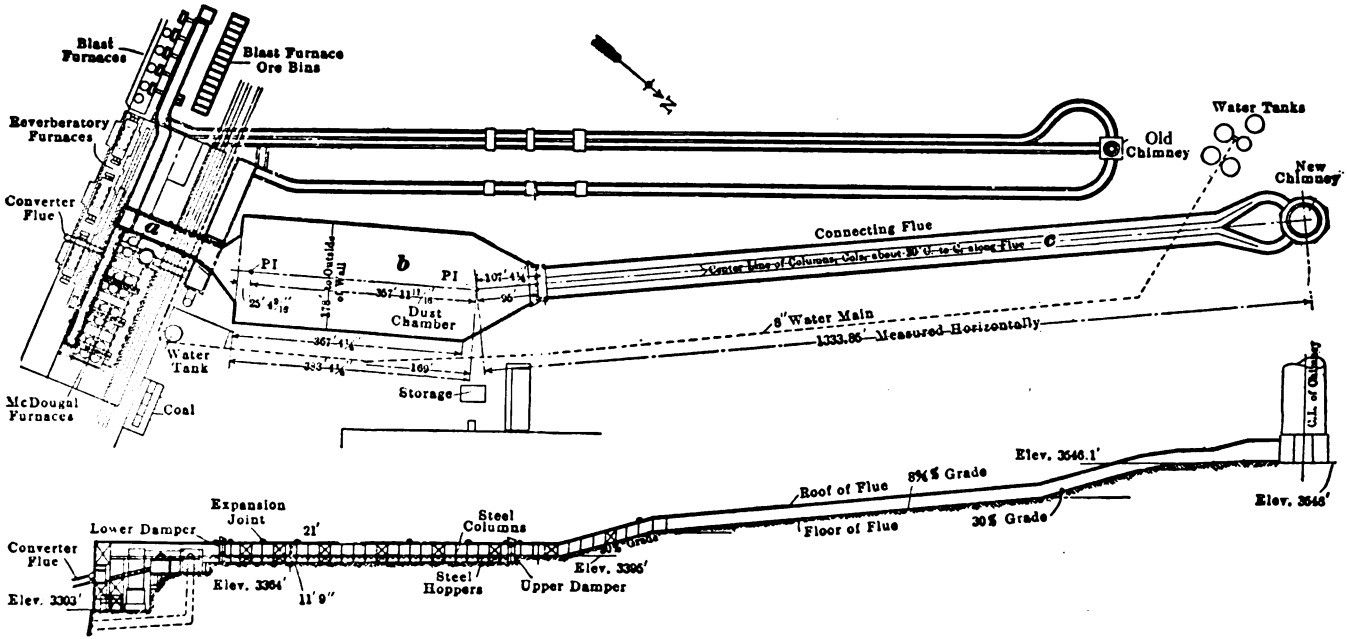


FIG. 111. — Location — plan and section of smelter and flue system

They give nearly as high a dust-collecting efficiency as any of these devices, and cause much less frictional resistance to the passage of the gases.

The filtering through cloth of the enormous volume of gases resulting from even a moderate-sized smelter would appear to be about as unpractical and unpromising a method of separating the fume and dust which they contain as could well be devised; yet long experience has taught that it is one of the most complete and useful methods at the disposition of the metallurgist, and

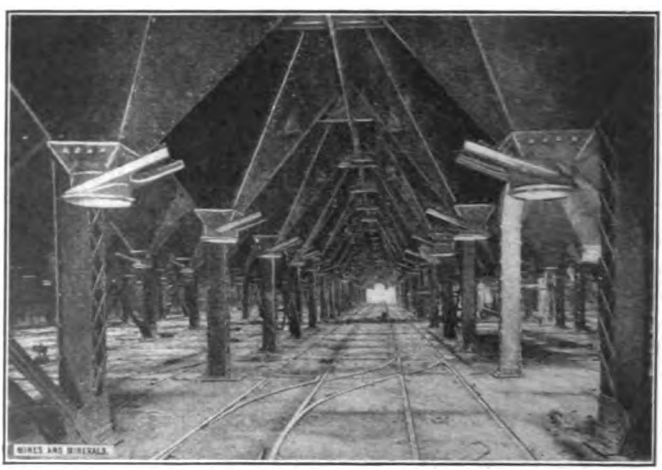


FIG. 112. — Basement, showing hoppers below dust-collecting chambers

may even be ranked as among the most important metallurgical advances of modern times.[1]

The bag-house has long been employed by lead and zinc smelters as the most suitable and efficient means of removing fume and fine dust from their furnace gases. In ordinary American practice the furnace gases are cooled, and their dust-content largely removed, by traversing long flues and chambers. Somewhere near the base of the stack an exhaust-fan removes these gases, and forces them, at the desired velocity and pressure, through a

[1] I regret that it is impossible in this volume for me to do more than make brief mention of the most salient features of this important subject. In doing this, I make free use of a paper by Prof. W. C. Ebaugh, in the *Journal Ind. and Eng. Chem.*, Oct., 1909.

great number of cotton or woolen bags which are suspended in the structure known as a "bag-house." This structure often contains 2000 to 4000 bags, each of which may be 30 ft. long and 18 inches in diameter.[1]

While cotton cloth — closely woven and having 42 to 50 threads to the linear inch, with a weight of 0.4 to 0.7 oz. per sq. ft. — has been largely used, it deteriorates so rapidly that the copper smelter finds it preferable to use the more expensive woolen bags. Erbaugh gives as the extreme of permissible temperatures for the gases, while using woolen bags, 21 deg. C. and 132 deg. C., the safest temperature being about 70 deg. C.

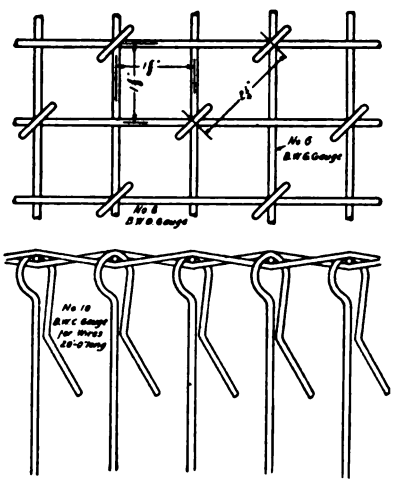


FIG. 113. — Method of hanging wires in dust-chamber

This method of filtration through cloth was found impracticable for the gases produced from the smelting of ordinary copper sulphide ores, owing to the rapid destruction of the bags, due mainly to sulphuric acid formed by sulphur trioxide which results from the breaking up of sulphates produced in roasting. A distinctly harmful effect is also produced upon the cloth by such sulphates as escape decomposition, though this cause is not nearly so important as the sulphuric acid. Rhodes and Sprague, in 1907, showed that it was possible to neutralize these acid substances without undue expense, and thus render the gases of copper plants suitable for filtering in a bag-house.

As the difficulty and expense of this neutralization method stand in close relation to the amount of sulphuric acid which the gases contain, it is evident that the gases from the roasting of pyritous copper ores are not so eligible as those from blast-roasting, blast and reverberatory-smelting furnaces, and converters, which contain a much smaller proportion of this destructive

[1] One foot (12 inches) = 0.3048 m.

substance. Nevertheless, the treatment has now been extended to the point where it is believed that even the roaster gases may be economically neutralized, and that the entire fumes from a smelter plant may thus be subjected to this treatment with advantage.

For neutralization purposes, the inventors make use of zinc oxide, lime, or magnesia. One of the great difficulties to be overcome was to effect a thorough mixing of the neutralizing

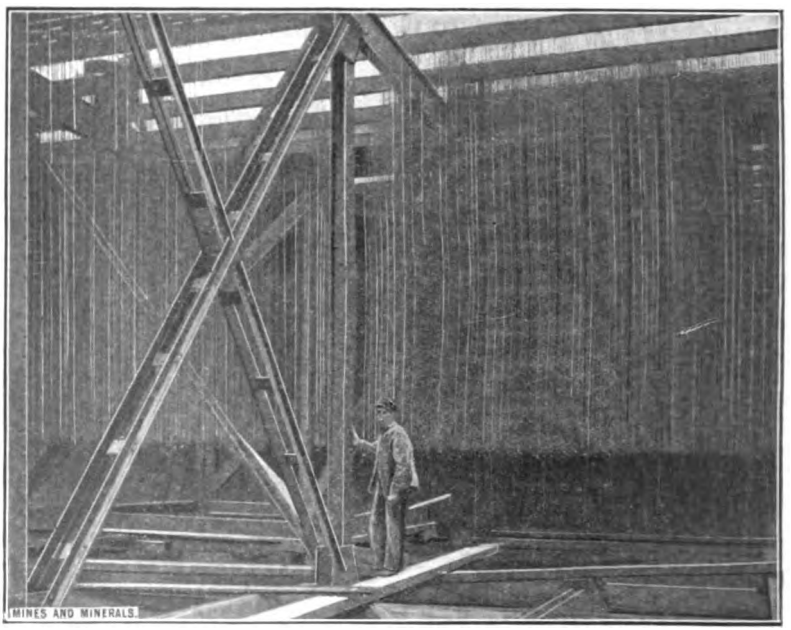


FIG. 114. — Interior of dust-chamber after hanging wires

agent with the flowing gases. Where the proportion of sulphuric acid is not too large, it is best neutralized by zinc oxide evolved in a vapor from zinc ores treated in special furnaces. Thus, the zinc oxide itself being in the condition of fume, an intimate mixture of the two streams of gas is easily effected. When sulphuric acid is present in larger amount, the greater portion of it is neutralized by the introduction of powdered lime or magnesia by means of special appliances, while this action is supplemented by the zinc vapors in order to make sure of complete neutralization. Theoretically 81 parts of zinc oxide, 56 parts of lime, or 40 parts

of magnesia are required to neutralize 98 parts of sulphuric acid; in practice, more than this is used for safety.

The amount of filtering surface required is determined more by the quantity of fume to be recovered than by the actual volume of the gases. The amount of fume[1] varies with the composition of the ore, and is often from one to two per cent of its weight. Ordinarily, at least 25,000 sq. ft. of filtering surface should be provided for each ton of fume produced per 24 hours.

(4.) Precipitating the fume by passing the gases through a space charged at numerous points with static electricity.

It is well known that, when gases contain suspended particles of solid or liquid (not gaseous) substances, and are brought into an electric field, there is a more or less complete deposition of the suspended particles. F. G. Cottrell has applied this principle to the treatment of smelter smoke and fumes with sufficient success to cause its adoption by one of the large lead-silver smelting companies, and it is believed that it will also be suitable for copper smoke under certain conditions.

The principal features of this application are:[2]

(*a*) The maintaining at suitable relative electrical potentials of two types of electrode surface;

(*b*) The charging of suspended particles by brush or glow discharge from the one type of electrode surface;

(*c*) The attraction of the thus charged particles to the other type of electrode surface;

(*d*) The prevention of the deposition of the suspended particles upon the surface of the solid insulating supports between the two types of electrode.

The Cottrell units are formed of steel chambers 10 ft. square with hopper-shaped bottoms provided with screw-conveyors to remove the deposited dust. The chambers contain a number of bars formed of two strands of asbestos and two copper wires set between a series of grounded iron plates, and are charged with positive electricity. The smoke passing through the apparatus is forced against the highly-charged, asbestos-covered bars, the

[1] C. B. Sprague in *Eng. and Mng. Journal* of March 8, 1910, p. 520.

[2] "Cottrell Process for Condensing Smelter Fumes," *Eng. and Mng. Journal*, Aug. 22, 1908.

solid particles receiving a positive charge which precipitates them on the low-potential iron bars.

3. *Gases.* — After eliminating all mechanically-suspended dust and all fume from the smoke of the copper smelter, the only injurious substances remaining that are of sufficient importance to be considered in this brief review are SO_2 (sulphur dioxide) and SO_3 (sulphur trioxide).

These gases do not, apparently, injure the roots of plants, nor do they damage the soil; their evil effect seems to be confined to the leaves, which wilt, acquire yellow or brown spots, and die. It is supposed that H_2SO_4 is formed by the oxidizing action of the vegetation itself. The blighting action is much greater in wet climates than in dry, and continuity heightens its destructive effect. There is no direct damage to animals.

Most of the injury to vegetation from smelter smoke, under conditions prevailing in the United States, emanates probably from the solid particles of flue-dust containing soluble metallic salts, and from the particles of fume. Where all this material is settled thoroughly and, especially, when filtered through bags, which necessarily presupposes the neutralization of its SO_3, the remaining SO_2 will work but little damage when discharged at a reasonable altitude. It will be seen, therefore, that dilution with air offers a simple means for obviating the evil effect of these gases, and one that is peculiarly efficacious in dry climates.

The only other proved method of controlling the SO_2 fumes that seems feasible for a large plant is to convert them into sulphuric acid. This is a very difficult and expensive operation, owing to the fact that, during the ordinary processes of copper smelting, the escaping gases are too dilute, and are contaminated with dust, fume, and fuel gases, and diluted with air to an extent that introduces the most serious complications.

Despite these difficulties the pyrite smelter of the Ducktown Sulphur, Copper, and Iron Company, of Tennessee, has succeeded in making acid from the $3\frac{1}{2}$ per cent SO_2 gases of its pyrite furnaces with the use of a chamber capacity of only 13 cu. ft. per pound sulphur burned in the ore. The daily production at present is about 160 tons of 60 deg. B. acid, which represents the fumes from one single blast-furnace.

The success of this enterprise does not mean, however, that

it would be possible for every great smelter of pyritous copper ores to convert his fumes into sulphuric acid. In most cases in the West of America there is no possible use for the acid. Unless there are contiguous oil-refineries that can utilize it, or phosphate-beds where it may be employed in the manufacture of superphosphates, or low-grade, silicious ores suitable for lixiviation, the acid is practically valueless.

CHAPTER XVII

THE purpose of this chapter is to present the subject of *Copper Slags* in a condensed form and in a manner which shall be useful to the practical smelter. In order to accomplish this, scientific considerations must be subordinated to utility, and exceptional conditions can receive but scant notice.[1]

In order, however, to render this discussion more useful to a large class of men who engage in mining and smelting without a thorough technical training, I shall begin the subject with a brief statement of its fundamental principles.[1]

I have already explained that the copper smelter finds it convenient to regard his furnace-charge as consisting of two distinct portions, although in reality these two portions are usually in a condition of intimate mixture. They are (*a*) metallic portion; (*b*) earthy portion.

The metallic portion consists of such constituents of the charge as will, on melting, yield matte (or even metallic copper in the rare case of smelting purely oxidized ores). Usually this portion is made up mainly of sulphides of certain metals, iron and copper predominating.

The earthy portion consists of such constituents of the charge as do not yield matte (or metal) on melting. This ordinarily comprises silica, iron oxides, lime, alumina, magnesia, etc.

The metallic portion of the ore melts into a completely liquid matte at a comparatively low temperature — inside of 1000 deg. C. — regardless of the proportion in which its chief constituents are present.

The earthy portion, however, will not melt at a suitable temperature unless its various constituents are kept within certain limits. As Nature seldom furnishes ores containing these earthy constituents in exactly the proper proportion to form an

[1] The reader is referred to "Principles of Copper Smelting" for a more extended study of slags.

[2] Consult page 25 et seq. in connection with the opening portion of this chapter.

easily-fusible compound, it becomes necessary for the operator to add what is lacking, or to remove what is superfluous.

In much of the very ancient metallurgical work — and indeed today amongst rude peoples — the simple plan is adopted of subjecting a copper ore at once to the operation of smelting, even though its earthy constituents are such that they will not melt into a slag at all. The easily-fusible metallic portion is melted and drained away from the infusible earthy lumps, and the latter are rejected without ever having been melted at all. This is merely an imperfect species of liquation, and — save in a few exceptional cases — is unworthy of serious consideration, for several reasons: for instance, the separation of the metallic and earthy portions is very imperfect, and the rejected unfused mass is too rich in valuable metals. Again, the removal of the unmelted earthy lumps is an arduous and expensive task, destroys the continuity of the process, and is only feasible at all when working on a very small scale.

Experience has taught that the only suitable method to pursue is to plan the earthy constituents of the charge in such proportions that they will melt together easily into a liquid slag, through which the heavier metallic globules can sink with freedom. By this operation two results are effected: (*a*) the metallic portion separates from the earthy portion; (*b*) the furnace clears itself automatically of its contents, and the smelting operation becomes continuous.

At present we have nothing to do with the metallic portion of the ore, and may concentrate our attention upon the earthy, or slag-forming portion.

It seems to me that in beginning the study of slags from first principles there are three especially prominent questions which force themselves upon our attention:

1. What qualities do we demand in any and every slag which we may be called upon to produce?

2. What are the more important substances with which we may be called upon to deal in producing slags?

3. What is the chemical and physical behavior of these substances — both alone and in mixture — at ordinary furnace temperatures?

1. *What qualities do we demand in any and every slag resulting from the smelting of copper ores?*

(*a*) *Liquidity;* so that it may be able to flow out of the furnace, and to permit the free sinking through it of the globules of matte or metal.

(*b*) *Specific gravity sufficiently low;* so that the matte globules may separate from it quickly and completely.

(*c*) *Fusibility;* so that it may melt at a temperature which is not beyond our power to produce economically.

(*d*) *Economy;* so that the foregoing essential qualities shall not have been produced at an unreasonable cost.

These four fundamental points are obvious to any person of ordinary capacity, and require no further elucidation.

2. *What substances are we called upon to deal with in producing slags?* Of the long list of elements which Chemistry has isolated, only about 20 occur in ores, fluxes, or fuels in sufficient quantity to have any practical bearing upon the operation of smelting copper ores. These 20 elements may, for present purposes, be divided into three groups:

Matte-forming Elements	Volatile Elements	Slag-forming Elements
antimony	antimony	aluminum
arsenic	arsenic	barium
copper	carbon	calcium
gold	hydrogen	iron
iron	lead	magnesium
lead	nitrogen	manganese
silver	oxygen	oxygen
sulphur	sulphur	potassium
zinc	zinc	silicon
		sodium
		zinc

The nine elements of the first group, variously combined, go into the matte.

The nine elements of the second group, variously combined, pass off as gases.

The eleven elements of the third group, variously combined, enter the slag.[1]

[1] Certain of these elements may go partly in one direction and partly in another; or may indeed, in part, go in all three directions. Thus zinc is inclined to go into the matte, to volatilize with the furnace gases, and to enter the slag. Consequently, I have felt obliged to include it in all three of the above groups. Copper, on the other hand, although quite ready (under oxidizing conditions) to combine with silica and enter the slag, is always

As we are concerned at present only with the slag-forming elements, we may drop further consideration of the first and second groups, while we make the third group the object of careful attention.

In the first place, we may see if there are any general laws which apply to all the members of the group. One fact in particular stands out prominently: all of the slag-forming elements enter the slag in an oxidized condition.[1]

The one element, therefore, which is present in every molecule of every element which normally enters the slag is oxygen.[2]

There is still another interesting feature which applies to almost all of the members of this slag-forming group: omitting oxygen, which is itself the oxidizer, we note that every element in the group, excepting silicon, becomes a base on combination with oxygen, whilst the oxidized silicon itself, at high temperatures, plays the part of a strong acid, combining with these bases to form salts called *silicates*. In order to appreciate as clearly as possible the phenomena attending slag-formation, we must conceive of silica as a strong anhydrous acid, having a powerful — though differing — affinity for each of the bases of this third group.[3]

protected by sulphur or carbon, and thus forced to enter the matte or metal. The minute portion of copper which is always found in the slag results partly from accidental globules of matte which have failed to separate properly. Even such copper as enters the slag in an oxidized form combined with silica is so slight in quantity that it has no effect upon the slag, and it would merely introduce useless complications if we attempted to include it among the slag-forming constituents of the ore. Every one of the twenty-odd elements just enumerated goes, to some extent, into each and all of the three groups; but if they were arranged upon such a basis, this chapter would contain very little that would be of use to the practical smelter. All through a book of this description it is necessary to temper strict scientific accuracy with common sense.

[1] I neglect, in this place, the presence of small quantities of sulphides in the slag. These are either mechanical enclosures, actual solutions of minute amounts of sulphides in the silicates, or salts which possess only a scientific interest.

[2] In the study of slags, I believe it to be more convenient to employ, at times, the older chemical notation in which an oxidized radical (base) is shown as combined with an anhydrous acid. Thus: calcium bisilicate is written CaO, SiO_2, the CaO being called the base, and the SiO_2 the acid.

[3] Under exceptional conditions, alumina (Al_2O_3), and even ferric oxide (Fe_2O_3), may play an acid rôle. It is not advisable to obscure the present discussion with these refinements.

We may, then, rearrange our third group slightly, in conformity with the statements just made. Oxygen may be dropped from separate consideration because it forms an essential part of every molecule in every slag-compound. Silicon may be placed in a separate division because its behavior is diametrically opposed to that of the remaining constituents, and all of the members may be regarded — and written of — as in the same condition of oxidation as that in which they exist when forming part of an ordinary slag. We have, consequently:

SLAG-FORMING SUBSTANCES

Acids	Bases
silica ..... SiO_2	alumina..... Al_2O_3
	baryta..... BaO
	ferrous oxide..... FeO
	magnesia..... MgO
	lime ..... CaO
	manganese oxide ..... MnO
	potash ..... K_2O
	soda ..... Na_2O
	zinc oxide ..... ZnO

The slag produced from the smelting of copper ores is invariably a silicate: that is to say, a chemical compound of silica and one or more of the bases just enumerated.

For convenience of description, silicates are classified according to the ratio between the amount of oxygen contained in their base side and in their acid side. The Latin prefixes sub, uni, bi, tri, etc., are employed to designate these artificial groups of silicates. The term subsilicate includes, of course, an infinite number of silicates — meaning simply a lower silicate — or one in which there is less weight of oxygen on the base side than on the acid side. For instance: $4FeO + SiO_2$ would be a subsilicate, as there are four atoms oxygen on the base side, and only two atoms on the acid side. A unisilicate (monosilicate, singulosilicate) has an equal amount of oxygen on both the base and the acid sides: for instance, $2FeO + SiO_2$, there being two atoms oxygen on each side. A bisilicate has two atoms oxygen on the acid side to one atom on the base side: for instance, $FeO + SiO_2$, thus having two atoms oxygen on the silica side to one atom on the base side. A trisilicate has, as its name implies, three atoms oxygen on the silica side to one atom on the base side: for instance, $2FeO +$

$3SiO_2$, thus having six atoms oxygen on the acid side to two atoms on the base side — which is the same as three to one.

Slags with so large a proportion of silica as the trisilicates are already too infusible for the conditions of the copper smelter so that it is not necessary to extend the nomenclature further.

These definite types of silicates may be regarded as milestones, or fixed guides, along the path of description; but they are somewhat far apart from each other, and there is one point where it is convenient to interpolate an additional member, thus halving the distance between two of the types. This point lies midway between the unisilicate and the bisilicate. The extra half-way type here introduced is called the sesquisilicate, there being one and one-half atoms oxygen on the acid side to one atom on the base side: for instance, $4FeO + 3SiO_2$, thus having six atoms oxygen on the acid side to four atoms on the base side, which is the same as one and one-half to one. But the sesquisilicate cannot be reckoned as an independent type, for every sesquisilicate may be resolved into a mixture of singulosilicate and bisilicate; thus, the sesquisilicate just given, $4FeO + 3SiO_2$ = one part singulosilicate $(2FeO + SiO_2)$ + two parts bisilicate 2 $(FeO + SiO_2)$, or, written in more correct chemical form, $Fe_4Si_3O_{10} = Fe_2SiO_4 + 2\ (FeSiO_3)$.

The following table brings all these type slags into a convenient form. The letter *R* is used to indicate any one or more of the bases (oxides) which are found in slags.

SILICATE-DEGREE

Name	Oxygen in acid	Oxygen in base	Ratio of O in acid to O in base, taking acid-O as unity		Ratio of O in base to O in acid, taking base-O as unity		Formula
Subsilicate[1]	1	2	1	2	1	½	4 RO + SiO_2
Unisilicate	1	1	1	1	1	1	2 RO + SiO_2
Sesquisilicate	6	4	1	⅔	1	1½	4 RO + $3SiO_2$
Bisilicate	2	1	1	½	1	2	RO + SiO_2
Trisilicate	3	1	1	⅓	1	3	2 RO + $3SiO_2$

[1] As the term subsilicates includes *all* slags which contain more O in the base than in the acid side, their number is, of course, very large. I simply select the one given in the table to serve as an illustration for this type of slag.

All the basic oxides (*bases*) have different molecular weights, and, therefore, combine in different proportions with silica; consequently, if we desired to calculate a polybasic slag, containing, for instance, ferrous oxide, lime, magnesia, etc., and wished to find out the silicate-degree of that slag, we should find it a very complicated undertaking if we had to figure out the proportion of each of these bases separately, and then dole out to each of them, in proportion to its molecular weight, the silica which belonged to each, before we could determine the silicate-degree of that slag.

As formidable, however, as metallurgical calculations may often appear at first sight, we always have some short cut by which the chief difficulties may be avoided, or greatly diminished. In the present instance, our short cut may consist in employing *the oxygen-ratio between the base and acid sides of the slag as a means of determining the silicate-degree of the slag.*

Thus — as will be seen from the foregoing table — the oxygen ratio of acid to base is 1 to 1 for the unisilicate, 1½ to 1 for the sesquisilicate, 2 to 1 for the bisilicate, 3 to 1 for the trisilicate, etc.

This employment of the oxygen-ratio in determining the silicate-degree of slags is highly useful to the metallurgist and, though inferior in some respects to the direct comparison of the equivalent (combining) weights of the constituents, is still so universal, both in the literature and the practice of metallurgy, that I believe it to be the most useful method to describe in a work of this nature.

In making use of the oxygen-ratio of a slag to determine its silicate-degree, I have always found it convenient to calculate the actual *pounds* of oxygen on the acid side and on the base side of a given slag — as determined by its analysis — the result, of course, showing the silicate-degree at once. An illustration will make this clear.

What is the silicate-degree of a slag having the following analysis?

SiO_2	48.1	per	cent
FeO	26.91	"	"
Al_2O_3	9.10	"	"
CaO	11.4	"	"
MgO	1.9	"	"
	97.41	"	"

On Silica Side

	lb.		per cent		lb. O
SiO_2	48.1	×	53.33	=	25.65

On Base Side

FeO	26.91	×	22.22	=	5.98
Al_2O_3	9.1	×	47.	=	4.28
CaO	11.4	×	28.6	=	3.26
MgO	1.9	×	40.	=	0.76
					14.28

Thus this slag contains 25.65 lb. oxygen on its acid side to 14.28 lb. oxygen on its base side, and its silicate-degree is $\frac{2}{1}\frac{5}{4}\cdot\frac{6}{2}\frac{5}{8}$ = 1.8. It has 1.8 pounds oxygen on the acid side to one pound on the base side, and is, consequently, a little less silicious than a bisilicate.

Percentage of Oxygen in

Silica, SiO_2	53.33
Ferrous oxide, FeO	22.22
Manganous oxide, MnO	22.6
Lime, CaO	28.6
Magnesia, MgO	40.
Baryta, BaO	10.46
Alumina, Al_2O_3	47.

Experience has taught us that almost all of the slags which are suitable for the purposes of the copper smelter lie between the unisilicate and the bisilicate: that is to say, between the type-slag which has one pound oxygen on the acid side to one pound oxygen on the base side, and that which has two pounds oxygen on the acid side to one pound on the base side. A slag more basic than the unisilicate is apt to form accretions, to carry off undue metal values, and to be unsuitable in various other ways; whilst a slag more acid than the bisilicate usually takes too high a temperature to keep it sufficiently liquid to flow freely.

Consequently, in planning a new slag, the metallurgist should always calculate its silicate-degree and make sure that it lies within the indicated boundaries. As a matter of fact, most ores — as mined or purchased — are much too silicious, and, if smelted without the addition of basic substances, would yield a slag more acid than the bisilicate, and totally unsuited to the purposes of the copper smelter. They require the addition of

basic ores, or basic fluxes, in order to reduce their silicate-degree to at least the standard of a bisilicate. Now, as the obvious means to reduce the silicate-degree is to add oxygen-carrying substances, it is plain that the base which contains the largest proportion of oxygen will be the most advantageous base to use, because a lesser weight of it will be required to introduce a given amount of oxygen.[1]

In other words, the smaller the equivalent weight of a base, the larger will be its content in oxygen, and the greater will be its replacement-value as compared with similar weights of other bases. This replacement-value of bases is of great commercial importance to the metallurgist, and is always taken into consideration in the calculation of furnace-charges. This point is shown clearly by the following illustrative table:

Bisilicates	Equivalent weight of base (oxide of the metal)	Percentage O in base	Percentage of base needed to form a bisilicate	Percentage of silica needed to form a bisilicate	Pounds silica that 1 lb. base will combine with to form a bisilicate	Pounds base that 1 lb. silica will combine with to form a bisilicate	Pounds base required to flux a constant amount of SiO_2
$MgO + SiO_2$ . .	40	40.0	40.0	60.0	1.5	0.667	1.0
$CaO + SiO_2$ . .	56	28.57	48.27	51.73	1.07	0.93	1.4
$FeO + SiO_2$ . .	72	22.22	54.55	45.45	0.833	1.2	1.8

We see from this table that — so far as lowering the silicate-degree is concerned — one pound MgO is as good as 1.4 lb. CaO, or as 1.8 lb. FeO. Consequently, if all these three bases could be obtained at the same price — say ten dollars per ton — ten dollars' worth of MgO would go as far toward lowering the silicate-degree of a slag as would fourteen dollars' worth of CaO, or eighteen dollars' worth of FeO. The commercial importance of this fact is too obvious to require illustration.

As this kind of calculation has constantly to be employed in making up smelting mixtures, tables have been prepared which

[1] It must be recollected that we are, as yet, only speaking of the *replacement-values* of bases, and have not yet begun to study the *fusibility* of the slags which these bases will produce. Consequently, no practical conclusions can be drawn until the question of fusibility has been settled.

save most of the figures. The following table by the late Professor Balling, of Bohemia, is in common use:

BALLING'S TABLES

One Part by Weight of Silica Requires:	Parts by Weight of Bases	One Part by Weight of Bases Requires:	Parts by Weight of Silica
For Unisilicates:		*For Unisilicates:*	
Lime	1.86	Lime	0.535
Magnesia	1.33	Magnesia	0.750
Alumina	1.14	Alumina	0.873
Ferrous oxide	2.40	Ferrous oxide	0.416
Manganous oxide	2.36	Manganous oxide	0.422
For Bisilicates:		*For Bisilicates:*	
Lime	0.93	Lime	1.070
Magnesia	0.66	Magnesia	1.500
Alumina	0.57	Alumina	1.747
Ferrous oxide	1.20	Ferrous oxide	0.833
Manganous oxide	1.18	Manganous oxide	0.845
For Sesquisilicates:		*For Sesquisilicates:*	
Lime	1.24	Lime	0.803
Magnesia	0.88	Magnesia	1.125
Alumina	0.76	Alumina	1.310
Ferrous oxide	1.60	Ferrous oxide	0.625
Manganous oxide	1.57	Manganous oxide	0.633

An illustration of the application of these tables to practical conditions may be of service.

Let us suppose that we intend smelting a silicious ore-mixture, with the production of a bisilicate slag. On calculating our slag-forming constituents we find that, after all of our bases present have been combined with the requisite amount of silica to form a bisilicate slag, there still remains 15 per cent of the total weight of our ore in the shape of excess silica. We can purchase various bases at the following prices:[1]

CaO	@	$7	per ton
MgO	@	9	" "
FeO	@	8	" "

[1] For the benefit of readers unaccustomed to the subject, I will point out that these bases are not obtained in an isolated condition, as might be supposed from the above quotations. The lime and magnesia would be purchased as limestone or dolomite, while the FeO would be derived from hematite, or — more probably — from pyrite.

Which of these bases should we purchase with a view of reducing our slag to the bisilicate degree at the least expense?

We have 15 per cent of excess silica to flux to the condition of a bisilicate. Fifteen per cent of one ton (2000×0.15) = 300 lb. Referring to Balling's tables, we find that 1 lb. silica requires 0.66 lb. MgO, or 0.93 lb. CaO, or 1.2 lb. FeO to form the respective bisilicates. Consequently, the 300 lb. excess silica would require

of MgO	300×0.66 = 198 lb. at	$9 per ton =	$0.89
of CaO	300×0.93 = 279 lb. at	7 per ton =	0.98
of FeO	300×1.2 = 360 lb. at	8 per ton =	1.44

Thus, to produce a bisilicate slag from a charge containing 15 per cent of excess silica would demand an addition, to each ton of the original ore, of eighty-nine cents' worth of MgO, or of ninety-eight cents' worth of CaO, or of one dollar and forty-four cents' worth of FeO.

3. *What is the chemical and physical behavior of the constituents of slags at ordinary furnace-temperatures?*

Having thus reviewed the elements of the constitution and calculation of slags, we may proceed to their actual formation in the smelting furnace, devoting particular attention to such types as are found most suitable for the reduction of copper ores.[1]

The formation of slags in the gradually rising temperature of the smelting operation may be approached in at least three different ways:

(*a*) We may select the comparatively neutral atmosphere of the reverberatory furnace, where the various constituents of the charge are in such intimate and prolonged relation to each other that — under the influence of the high temperature — they are forced to combine, or at least to effect some sort of mutual solution. Owing to these conditions, the true affinities of the ore-constituents are obscured, they being — as one might say — bullied by propinquity, and by extraneous heat, into results unconformable to their natural instincts. Consequently, reverberatory slags are the least spontaneous, and most artificial, of any slags, and are the least suited to use as types for the study of slag-formation.

[1] See "Principles of Copper Smelting" for a more extended study of the elementary part of this subject.

(*b*) We may select the strongly reducing atmosphere of the coke blast furnace, with its column of ore and fuel resting upon a foundation of incandescent fragments of coke, permitting no slag to escape until it has run this gauntlet and been modified in accordance with the chemical laws which prevail in the presence of carbon at a temperature of 1500 deg. or 1600 deg. C. This condition I believe to be more suited to the practical study of the formation of copper slags than is condition "*a*," but not nearly as instructive as the following situation, in which the slag-forming constituents of the charge are left free to work their own pleasure, modified in the smallest degree possible by outside influences.

(*c*) We may select the neutral (finally, intensely oxidizing) atmosphere of the pyrite furnace as the theatre of action in which to study the formation of slag. This, on the whole, seems to me the most suitable and enlightening field in which to pursue these studies, and the subject has been treated [1] at considerable length in Chapter VIII.

It is shown there that, under conditions prevailing in true pyrite smelting, the FeO, which results from the bessemerizing of the iron sulphide with a cold blast and under unfettered conditions, unites with free silica to form a silica salt (slag) whose formation-temperature corresponds to the degree of heat which reigns in the focus at that moment. As it has been demonstrated by experiment that the formation-temperature of ferrous silicates becomes lower as their content in silica increases (up to a point beyond anything that need concern us at present), we should expect that an increase of heat in the focus of the furnace would be accompanied by an increase in the iron content of the slag, and a corresponding decrease in its silica; this presumption is correct, as demonstrated by the results of practice.

In pyrite smelting this increase of heat in the focus can only be effected by burning sulphides more rapidly, which, in turn, can only be accomplished by forcing a greater volume of blast through the charge and arranging the construction and manipulation of the furnace to harmonize with the new and more forceful conditions.

It is found that when the true pyrite furnace is running well

[1] See the chapter on "Pyrite Smelting" in "Principles of Copper Smelting" for a more detailed study of the entire pyrite process.

and vigorously on its typical charge it will choose to form a tolerably basic slag, running down in silica as low as 30 per cent, and approximating closely the unisilicate type. For instance, the Mt. Lyell slag a few years ago, when the furnace was running on an almost typical pyrite mixture, is reported by Sticht as approximating the following composition (small amounts of Al_2O_3 and BaO being counted as CaO for this especial illustration):

SiO_2	31 per cent = 16.5 oxygen	
FeO	53 per cent =	11.77 oxygen
CaO	16 per cent =	4.6 oxygen
		16.37

This slag contains practically equal weights of oxygen in both base and acid, and is thus a unisilicate.

An important practical point, in attempting to plan one's furnace-charge in accordance with the melting-point of silica compounds, is to bear constantly in mind that few things in metallurgy are so deceptive and so dangerous as the formation-temperatures of slags. To the inexperienced metallurgist there is an almost irresistible fascination in a well-ordered table of the formation-temperatures of the variously-proportioned compounds of silica, ferrous oxide, and lime which have been so carefully determined by competent observers. There is nothing in literature more delightful to the superintendent swamped with silicious ores than to read such determinations as the following (Hofman):

SiO_2 Per Cent	FeO Per Cent	CaO Per Cent	Silicate-Degree	Formation-Temp. Deg. C.
31.47	45.68	22.85	1	1190
36.47	42.36	21.17	1.25	1180
40.80	39.46	19.74	1.50	1160
44.55	36.97	18.48	1.75	1140
47.86	34.77	17.37	2.00	1120
50.82	32.78	16.40	2.25	1115
53.44	31.04	15.52	2.50	1110

In these reliable and valuable determinations the fusibility of the slag improves steadily, as its silica increases, in a manner and to an extent that would promise to relieve the smelter of a

full half of all his embarrassments and put a premium on all silicious ores, were it not for the one unfortunate circumstance that should be emphasized by every metallurgical author whenever he has occasion to indicate the formation-temperature of a slag in a practical treatise: namely, that the *formation*-temperature of a slag stands in no definite relation to its *flowing*-temperature, nor does it throw much useful light upon its suitability for actual furnace conditions.[1]

In other words, a slag may have a very low formation-temperature, and yet will be so viscous that it will not flow out of the furnace at all; while, on the other hand, another slag may have a comparatively high formation-temperature, and yet will be exceedingly liquid, will flow freely out of furnace and forehearth, will permit a clean separation of the matte, will demand but a moderate amount of fuel, and will be satisfactory and economical.[2]

While it is convenient, and probably even essential, for technical purposes, to express the composition of our slags by the mere results of a chemical analysis in which the slag is represented

[1] For instance, it will be seen from the above table that the slag lying half-way between the bi and trisilicate, and containing 53 per cent silica, has a formation-temperature 50 deg. C. lower than the sesquisilicate, with 41 per cent silica. Yet every smelter knows that he begins to feel the presence of the enemy from the moment that the slag passes the 40 per cent silica-mark; he knows that it becomes more and more difficult to fuse as the silica rises, and that at 45 per cent silica his furnace-men have to be well trained to escape trouble, while about 48 per cent silica marks his limit for any steady continuous work, and 53 per cent is preposterous. He knows also that his fuel bills — be it in the shape of coke or in the form of sulphide-fuel debited to the furnace — will increase as his silica increases. The fact that the 53 per cent-silica slag has a lower formation-temperature than any of the others is of little comfort to him if it freezes his furnace in a few hours.

[2] Every metallurgical work that descends to the consideration of practical details ought to contain two sets of tables relating to slag temperatures. One set should be confined to the more strictly scientific portion of the text, and should exhibit the *formation-temperature* of the various compounds of silica and bases appropriate to the subject: that is to say, the temperature at which a properly-fluxed mixture of the slag-forming substances will combine and show incipient fusion. The admirable determinations of Hofman, and other investigators, have provided us with much of the necessary information on this point.

The second set of tables, intended especially for the practitioner, should indicate the *flowing-temperature* of the different silica salts which might be

as consisting of so and so many per cent of SiO_2, FeO, CaO, etc., we derive therefrom a singularly and unfortunately distorted idea of its actual make-up.

If we desired to describe the nature of the ore from a certain mine, consisting say of quartz, heavy spar, pyrite, and copper glance, we should obtain little idea of its real nature from the following analysis:

SiO_2	52	per	cent
BaO	10	"	"
Fe	12	"	"
S	18	"	"
Cu	5	"	"
	97	"	"

This analysis would be useful in calculating a smelting mixture, but would fail to offer the slightest picture of the true composition of the ore before it was smelted.

In the same way we have learned, of late years, that chemical analyses alone do not tell us all that we need to know about slags, and that the road to further knowledge in regard to their constitution — and consequent improvement in their general management — lies in much the same direction as was indicated in the preceding illustration: namely, in determining the minerals which the slag represents rather than the mere percentage of the various bases and acids that it may contain.

Attempts have long been made, with varying success, to

considered by the metallurgist as possible for his slag; these remain yet to be constructed. By the flowing-temperature I mean the degree of heat that is required to place a slag in such a condition that it will flow properly out of the furnace without chilling the tuyeres unduly, that it will allow metal globules to settle out of it, and that it will behave generally in the reasonable and suitable manner expected of a slag that is to be used for commercial smelting.

That class of determinations cannot well be made in the laboratory, as *volume* is an essential feature in the behavior of slags, and time — often extending over several days — is required to judge properly regarding the many details which go to make up the commercial suitability of a slag. Instead of such tables I can offer only a general statement of the extremes to which it has been found advantageous, in actual practice, to push the various more important slag-forming constituents, together with such brief descriptive remarks as may seem useful in explaining or modifying a bald statement of facts.

identify at least the crystalline slags with natural minerals. The researches of Vogt have, however, been more extended and systematic than those of any former observer, and it is impossible to discuss the constitution and behavior of these artificial silicates without the light which is thrown upon the subject by the realization of the fact that practically all of our slags — crystallized, crystalline, and amorphous — have been identified with definite mineral species.[1]

With unimportant exceptions, all of the minerals found in slags are salts of one or the other of the two well-recognized silicic acids: namely, ortho-silicic acid, H_4SiO_4 or $2H_2O{\cdot}SiO_2$, and the meta-silicic acid, H_2SiO_3 or $H_2O{\cdot}SiO_2$. The ortho-silicate corresponds to the unisilicate of the metallurgist, while the meta-silicate corresponds to his bisilicate.

Upon calculating the oxygen-ratio of any given slag, it is found usually that it does not correspond exactly to a uni or a bisilicate, but — in the vast majority of cases — lies somewhere between these two silicate degrees. In such cases, the slag is a mixture of ortho-silicate and meta-silicate minerals. The exact species of minerals of which it is formed will, of course, depend upon the bases present, but the mineral entities in the slag will be definitely unisilicates and bisilicates.[2]

Vogt states that practically all metallurgical slags are definite mineral compounds or mixtures of minerals, the mineral or minerals found being solely a function of the silicious magma produced in the furnace.

The minerals are not always perfectly pure, their proper bases being frequently replaced to some extent by other bases, without destroying the individuality of the mineral; this replacement, however, must remain within certain limits, else the mineral form changes.

[1] J. H. L. Vogt, "Studier over Slagger," Bih. till K. Svenska Vet. Akad. Handlingar, 1884. "Beiträge zur Kenntniss der Gesetze der Mineralbildung in Schmelzmassen, u. s. w.," Archiv for Mathematik og Naturvidenskab, Christiania, Vols. XIII, XIV, 1888–90. "Die Silicatschmelzlösungen, I. Ueber die Mineralbildung in Silicatschmelzlösungen; II. Ueber die Schmelzpunkt-Erniedrigung der Silicatschmelzlösungen," Christiania, 1903–04. I have also made free use of Fulton's excellent chapter on slags in his "Principles of Metallurgy."

[2] Exceptionally, an actual sesquisilicate occurs, apparently formed by the hypothetic acid $H_8Si_3O_{10}$ or $4H{\cdot}_2O{\cdot}3SiO_2$.

Vogt finds also that the mineral or minerals crystallizing out of a molten slag-magma will depend upon the chemical composition of the magma; if this composition corresponds closely to that of any of the regular slag minerals, this mineral alone will result; if it does not so correspond, two or more of the minerals will form. In the accompanying illustration (from Fulton) is shown the relationship between mineral composition and chemical composition as regards both the degree of acidity and the bases present. That mineral crystallizing out first is mentioned, though it may be subordinate in amount, and be followed by another mineral in much larger amount.

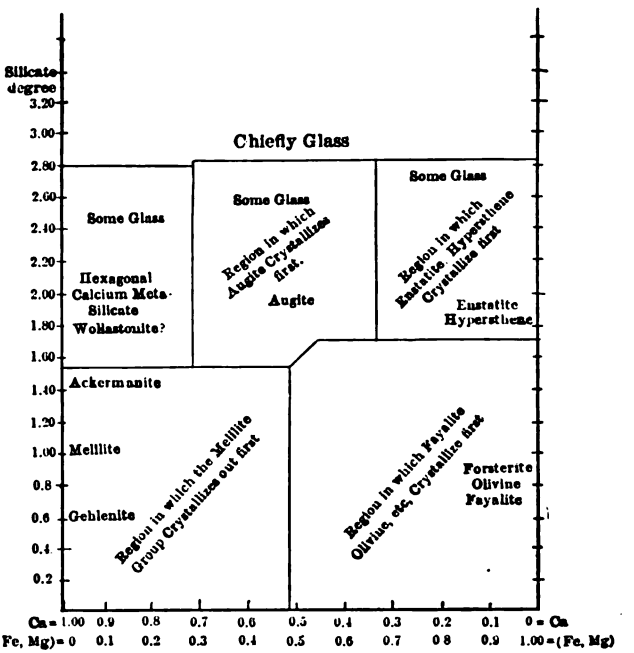


FIG. 115. — Diagram relating the mineral and chemical composition of Ca, Mg, Fe silicates in slags

If the magma has a silicate-degree of over 1.7, the first mineral to crystallize out will be a meta-silicate; if below this figure, the first mineral to crystallize out will be an ortho-silicate. In the presence of a certain amount of Al_2O_3, this limit may be lowered to 1.5.

The particular ortho- or meta-silicate that will first crystallize

out of a given magma depends upon the bases present, CaO to a certain extent opposing (Mg, Fe, and Mn)O.

Under certain conditions, aluminates may form such as spinel ($RO \cdot R_2O_3$), chiefly as aluminate. These conditions are, in the main: (*a*) the presence in the magma of more Al_2O_3 than can enter into the silicate minerals which form; (*b*) the presence of Mg or Zn, the latter being peculiarly conducive to the formation of spinels. In such a mixture the silicate minerals will appropriate such Al_2O_3 as they require, and the excess will separate out as spinel, which crystallizes out first and at a high temperature — a peculiarly undesirable feature.

On this basis it is seen that the mineral entities which crystallize on cooling were in solution as such in the silicate magma, and that slags must be classed as physical mixtures of the nature of alloys. The time required to cool, the temperature to which the molten mass is heated, etc., are stated to play no part in the determination of what the minerals are to be which will result from a given magma.

Beyond the bisilicate degree, definite mineral compounds do not form on cooling, the slag congealing mainly to a glass (solid solution).[1]

"Hitherto, the slags have been considered as having already been formed, *i.e.*, are in the condition of molten silicate magmas which, on cooling, form certain mineral entities. In actual furnace operation the question must be looked at from a somewhat different viewpoint. The furnace-charge will consist of, not the formed silicates, though some of these may be present as silicate minerals in the ore, but rather as the individual bases, such as CaO and MgO in the form of limestone, FeO in the form of Fe_2O_3 and Fe_3O_4 in oxidized or roasted ores, and FeS, FeS_2, etc., in sulphide ores. These bases, with the quartz, SiO_2, of the ores, form a mechanical mixture, and are so proportioned by calculation that they smelt together into the slag discharged eventually from the furnace. It is a well-known fact that the temperature necessary to cause these various bases and the silica to unite to form a slag is different from the melting or solidification temperature of the formed slag. The temperature necessary

[1] The following quotation from Fulton's "Principles of Metallurgy" deals so succinctly with some important practical applications of these views that I give it at some length.

to cause the various ingredients of the final slag to unite chemically and go into solution is known as the 'formation-temperature' of the slag. This formation-temperature, for any given mechanical mixture, is a function of the size of particles comprising the mixture, and of time.

"It has been shown by Doelter that in any complex mixture of bases and silica subjected to such a temperature as to cause fusion, certain silicates, namely, mineral entities, will form first, *e.g.* those which have the lowest formation-temperature; and that this mineral will absorb the rest of the mixture gradually as the temperature rises, until the whole is in combination or solution. The formation of any given mechanical mixture may be appreciably lowered by adding to it a certain percentage of formed slag of the same chemical composition, thus not changing the composition of the whole; for, from what has been stated above, the 'slag' will have a lower melting-point than the corresponding mechanical mixture, and the added slag melting first will aid materially in the rapid solution and combination of the ingredients of the mixture. It is a well-known fact of furnace practice in lead and copper smelting that the charge should contain a certain amount of slag as such; not only because this slag in some cases needs to be resmelted for its contained values, but also because it aids greatly in the operation of the furnace, probably for the reason above stated.

"The viscosity, or resistance to flow, of slags is of as much importance from a metallurgical standpoint as is the melting-point. Many slags of low melting-point are so viscous at that temperature that they will not flow freely from the furnace; in fact, may not flow at all. No exact standards of fluidity have been applied to slags; but making the subdivisions of viscous-fluid and thinly-fluid, it may be said that the degree of fluidity of a slag is a function of chemical composition and of temperature. With any slag of given composition, fluidity increases with a rise of temperature; but this increase of fluidity is by no means the same for different slags for the same increase of temperature.

"From experience, it may be stated that:

"1. The viscosity of slags increases, other factors being equal, with the degree of acidity (*i.e.*, silicate-degree) above the orthosilicates. The more acid silicates are decidedly more viscous

than the more basic silicates at the same temperatures (referring to melted slags).

"2. As regards bases present, FeO decreases the viscosity, and MnO acts in the same direction with still greater intensity. In the Ca-Mg silicates with some Al_2O_3, *viz.*, blast-furnace slags, those high in Mg are more thinly fluid than those high in lime. The effect of MgO, however, is not nearly so pronounced as that of FeO or MnO. In basic slags — *i.e.*, low silicate-degree — the chief base being CaO, Al_2O_3 has the effect of increasing the fluidity; but in acid slags the effect of Al_2O_3 is markedly to increase the viscosity, when it amounts to more than 10 per cent. From the glass industry, it is known that the presence of much K_2O and Na_2O in acid lime-silicates decreases the fluidity, the effect of the former base being the more pronounced."

Certain of the sulphides — notably calcium sulphide — dissolve in slags to a considerable extent. Upon cooling, this sulphide usually separates in minute crystals which, in very basic slags, may often be seen in abundance. The more acid slags are but feeble solvents of sulphides, while with basic slags high in zinc, the dissolved ZnS may run as high as six per cent of the weight of the slag. FeS is less soluble while, fortunately, Cu_2S is dissolved but slightly. One reason why it is impossible, even with the most complete settling, to get a basic slag as low in copper as an acid one is because the former slag has a much greater power of dissolving cuprous sulphide.

It seems to me that, in a book which is designed mainly as a guide for the practising metallurgist, the most convenient way in which to take up the study of practicable slags will be to consider separately each of their more important constituents, and to examine the highest and lowest extreme to which each constituent may be pushed in actual work, as well as to become familiar with the happy medium at which its advantages have their maximum development, and its drawbacks their minimum influence. The space allotted to this subject is too brief to permit of much discussion, or to warrant much consideration of exceptional conditions.

Silica.—This is the only universal constituent of copper slags, and is of the first importance in two separate ways: (*a*) it is

indispensable as the acid member of the fusible salt that we desire to form; (*b*) it is usually present in our furnace-mixtures in excess, and our commercial success may depend largely upon our skill in dealing with this unwelcome excess.

Our knowledge of the behavior of silica, and its compounds with our ordinary bases, has been acquired by long experience under an enormous variety of conditions, and could be stated in terms sufficiently precise to have a great value to the practitioner, were it not for the fact that it is frequently associated with another oxide which replaces, or modifies, or obscures the behavior of the silica in so marked a manner as to increase seriously the difficulty of the subject for the author, and still more for the metallurgist.

I refer, of course, to alumina which, for at least two generations, has been regarded as playing at least four different rôles by corresponding classes of observers, while the greater proportion of metallurgists had no fixed views upon the subject, excepting that alumina above six per cent was pretty sure to make its presence known in the furnace. The four commoner theories as to the behavior of alumina to which I refer are: (*a*) that it always takes the part of an acid; (*b*) that it always is a base; (*c*) that it may be either a base or an acid, according to the composition of the slag; (*d*) that it is to be considered as a neutral and indifferent substance, uncombined, and merely held in solution by the slag.

Personally, I have been unable to form an opinion, my own experience — as well as the recorded observations of others — having been so contradictory as to confuse, rather than enlighten, me.

The extended observations of Vogt indicate that all of the four classes of partisans may have been correct in their views to a certain extent, and, while it is impossible to appreciate the full significance of his discoveries without more complete information than I have been able to give in the preceding pages, I must at least cite his conclusions briefly, as I find in them the first explanation as to the behavior of alumina in slags that seems to me reasonable and capable of covering all situations. Future observers may extend or modify these conclusions, but, for the present, I believe them to be more useful to the metallurgist, and resting upon a more satisfactory foundation than any others that have been offered.

Leaving the more extended discussion of alumina to the sec-

tion devoted to the consideration of that oxide, I will merely state that the observations already referred to show that alumina enters into combination as a base to such an extent as is necessary to form the especial mineral, or minerals, that correspond to the chemical composition of the slag under consideration, and that any excess alumina that remains after these minerals are formed will behave as an acid, forming spinel — especially with MgO and ZnO. If the bases to form these aluminates are not present, the excess of alumina remains apparently in solution. As the spinels are very infusible, it is easy to see how refractory a slag may become when its content in Al_2O_3 is not present in the right proportion to join as a base in the formation of one of the more fusible silicates, but combines with MgO or ZnO to produce the infusible aluminate.

It must also be noted that, even where Al_2O_3 plays the part of a base, it has a strong tendency to raise the flowing-temperature of minerals and that, even under these favorable conditions, the viscosity of a slag will increase materially when it contains more than six or eight per cent of alumina, and the proportion of fuel will have to be raised in order to obtain a temperature sufficiently high to overcome this viscosity.

Returning to the consideration of silica, it may be said that the metallurgist dislikes, as a rule, to produce a slag below the unisilicate degree in smelting copper ores. Slags so low in silica have, contrary to general belief, a decidedly higher formation-temperature than the more silicious ones; but, being thin and fiery, they give the impression of great fusibility. They chill easily and tend to form accretions in hearth and settler, unless so much fuel is used that they are superheated and cause cutting and corrosion.

If their principal base is ferrous oxide, their specific gravity is high, and the settling of the matte becomes more difficult, while they take into actual solution more Cu_2S than do the more acid slags.

Such a slag would only be made where there was a deficiency in silica; because, in most districts, bases — and, especially, FeO — are far too precious as a flux for silicious ores to be wasted in producing a slag any more basic than is absolutely necessary for rapid and economical smelting.

The unisilicate is about the type of slag, for instance, that results from true pyrite smelting of the rapid and vigorous kind: that is to say, with cold wind, high ore-column, large volume of blast, free silica with a minimum of combined silica acting as a dead weight, much massive pyrite in the charge, with merely the 10 or 12 per cent of earths essential to a favorable slag. Such an instance is the type of slag produced (spontaneously) at the Mount Lyell furnaces a few years ago; it approximates the unisilicate, and has about the following composition:

SiO_2	31	per cent
FeO	53	" "
Earths, including Al_2O_3	16	" "
	100	" "

Excepting in pyrite smelting — where the iron is in great excess, and the main object of the process is to slag it with as little silica as possible — such a slag would be exceedingly rare, for the simple reason that iron is too valuable a base, as a promoter of fusibility, to waste it in such a reckless manner. Consequently, far the greater proportion of copper slags are governed by two limiting factors (assuming always that iron is scarce and that silica is present in excess): (*a*) they must contain as low a proportion of base — and as great an amount of silica — as is consistent with rapid and economical smelting; (*b*) this proportion of base, in its turn, must consist of as small a proportion of FeO as is compatible with good results.

This brings us again to the subject immediately under discussion: namely, the extreme limit to which it is found wise to push the silica content of an ore-slag. We have already learned that, except in unusual cases, it is not advantageous to let the slag fall below the unisilicate, the actual percentage of silica in the slag depending, of course, upon the equivalent value of the base which may happen to be present, and ranging from about 29 per cent silica in the pure ferrous silicate up to perhaps 33 per cent silica in cases where the FeO is replaced, to a considerable extent, by the earthy bases. This is decidedly too basic a slag for comfortable furnace conditions or for rapid and complete separation of matte globules.

The opposite extreme is the one which is the more important to the great majority of metallurgists, owing to the excess of

silica and the deficiency in iron which are the ordinary conditions that we have to meet in practice. It may be said, in general terms, that this high-silica mark is limited by the bisilicate degree; consequently, it is easy to remember that our practicable slags all lie between these two great natural boundaries — the unisilicate for the low extreme of silica, and the bisilicate for the high extreme. This statement approaches so closely to absolute correctness that it may be regarded as final, excepting for the few experts whose skill and experience embolden them to overstep the boundaries slightly in extraordinary cases, and who will not require the aid of a treatise on the elements of metallurgy.

The nearer we approach either of these extremes, the greater the difficulties become in the management of the process, and, conversely, the farther we recede from them, the more comfortable and satisfactory do we find the operation of the furnace and the result of the fusion.[1]

Ease of smelting and low slag-losses may be purchased at too high a cost. The aim of smelting is to produce the greatest profit that is possible under the circumstances, and the metallurgist is usually forced to forego the satisfaction of making a perfect slag, and content himself with one that is distinctly inferior, but that yields a larger profit under existing conditions. Smelting is almost invariably a compromise process, and the most successful metallurgist in practice is not the one who aims to produce an ideally perfect slag, but rather the man who knows how to select the imperfect slag that will yield the most profitable results.

As the percentage of silica in a slag of given silicate-degree varies with the replacement-value of the bases which it contains, it is impossible to continue discussing generalities, and we must now narrow the field. Hitherto, we have referred mainly to the *silicate-degree* of a slag; or, if we have spoken at all of the relative advantage of the different bases at our disposition, we have considered merely their *replacement-value*, and have not studied the important practical point of the fusibility and suitability of the resulting silicate.

For instance, one pound of CaO has a replacement-value (that is to say, will go as far in lowering the silicate-degree) equal to 1.285 lb. FeO. As lime is, under ordinary smelting conditions,

[1] This last remark must be regarded as a somewhat general statement, subject to certain moderate reservations and exceptions.

much more easily and cheaply obtained than FeO, it follows that, if the metallurgist were not restrained by some powerful consideration, he would never add iron purely as a flux for silica, but would confine himself exclusively to the cheap and plentiful limestone. He is, of course, restrained by the fact that a slag cannot be made too high in lime and too low in iron without becoming so infusible that it is unsuited to his conditions.

This fact demonstrates then that while, for *commercial* reasons, we must always bear in mind that as a means of lowering the silicate-degree of a slag CaO is worth 1.285 times as much as FeO, and MgO is worth 1.8 times as much as FeO, and Al_2O_3 is worth 2.1 times as much as FeO, we must take care, for *metallurgical* reasons, that we do not attempt to replace FeO by these more advantageous bases to any greater extent than will yield a slag that is fusible and suitable.

Recollecting that we are still discussing the extreme high and low limits to which silica may be pushed in commercial slags, we may complete the review by saying that — speaking from the standpoint of silicate-degree — the bisilicate represents the one extreme, and the unisilicate the other extreme, for practicable slags under any ordinary conditions.

If it is required to express the silica limits in actual percentages, I should say that these might be placed, respectively, at 30 per cent and 50 per cent, the latter extreme being attainable only under exceptional conditions, demanding unusual care and skill, and accompanied by slow smelting and high fuel-consumption.

Ferrous Oxide and Manganous Oxide.[1] — No slag, in actual practice, is ever a pure silicate of a single base. In the most extreme cases there will be some five per cent of other bases, derived from coke-ash, cleanings, country rock, and the minute quantities of CaO, MgO, Al_2O_3, ZnO, PbO, etc., which accompany even the purest ores. Recognizing this fact, it may be said that FeO is unique in being the only base that is metallurgically self-suffi-

[1] In this brief review it will be convenient to assume that MnO may replace FeO to any extent without changing materially the qualities of the resulting slag. It is not only that these two kindred bases have almost the same replacement-values (equivalent weight of FeO = 72; equivalent weight of MnO = 71); but the behavior of their silicates at furnace temperatures is so similar that it is safe to add them together and rate them both as FeO.

cient: by which I mean that it is the only base which, when combined with silica, forms a practicable slag.

From the degree of unisilicate (with about 70 per cent FeO and 30 per cent SiO) up to that of bisilicate (about 54 FeO to 46 SiO_2), it forms a series of fusible slags, the formation-temperature of which becomes steadily lower as the proportion of silica increases, until, at the bisilicate extreme, it melts at the low temperature of 1110 deg. C., which is 160 deg. C. lower than the fusion-point of the unisilicate. This behavior illustrates typically the danger of judging of the suitability of any slag by the mere consideration of its formation-temperature. In actual practice such a high silicate-degree for a pure (95 per cent) ferrous silicate slag is very far from advantageous.

Beginning at the unisilicate end (which corresponds to about 30 per cent SiO_2, 65 per cent FeO, and 5 per cent miscellaneous bases), we find no especial difficulty with this highly ferruginous slag excepting the inconveniences inseparable from such heavy basic silicates, which have been already considered. This slag is, naturally, extremely heavy and corrosive, requires extended settling facilities, and will hold in actual solution sufficient Cu_2S to cause a perceptible, though small, loss. For these reasons it is entirely unsuited to reverberatory work, and is far from welcome even in its natural habitat — the blast-furnace.

With increasing silica, and decreasing iron, its qualities improve steadily until it reaches about 38 per cent silica, 57 per cent FeO, and 5 per cent miscellaneous bases. Somewhere near this point, it attains its most admirable qualities and corresponds nearly to a 3 to 4 silicate (three parts oxygen on the base side to four parts oxygen on the acid side). It is not quite so free from metal values as though it contained a few per cent more SiO_2, and, in the reverberatory furnace, it cuts into the fettling slightly more than it would if it were a little more acid; but these slight disadvantages are outweighed by the rapidity and regularity with which the smelting operation proceeds, and by the general comfort which pervades the process when running upon a slag of this nature.

From this point, an increase in the proportion of silica begins to make itself felt, although it may reach 40 per cent before it becomes so unmistakable as to demand some slight change in the management of the process. If no measures of this nature are adopted, the furnace will fall off in capacity, the tuyeres will

require more frequent punching, the hearth will begin to build accretions, the slag stream will lose its white-hot appearance and will exhibit a slight viscosity, and the settler will show a tendency to chill rather than to cut. All these phenomena are signs of a lowering vitality, and are due to the fact that the more silicious slag, though having a lower formation-temperature, has a distinctly higher flowing-temperature than it did before its proportion of silica was increased. In other words, there is not sufficient heat within the furnace to melt the present charge rapidly and thoroughly, and measures must be taken to raise this interior temperature 50 or 100 degrees, or whatever may be found necessary, to restore the process to its normal condition of activity. This is effected, of course, by burning more fuel per unit of time. Whether this fuel shall consist solely of coke, or solely of sulphide, or of a little of each, depends upon our ore-mixture and upon the nature of the process itself.

These last statements refer to blast-furnace practice; but, if we are using the reverberatory, we shall still be confronted by the same conditions, and must still make use of the same remedies. The higher flowing-temperature of the more silicious slag demands a higher furnace temperature, and this, in turn, demands either that we burn more pounds of coal per minute per unit of grate area, or that we procure fuel having a higher calorific power.

It would be unprofitable to follow the behavior of the slag minutely with every increasing per cent of silica. Suffice it to say that, while it is practicable in both blast and reverberatory furnaces to raise the silica of a pure (95 per cent) ferrous silicate slag up to, or even a little beyond, the bisilicate limit (say even to 47 per cent silica), it becomes, above 42 per cent silica, so viscid and so demanding of increased fuel that it is usually more profitable to add sufficient lime, or other base, to keep the silica below this limit, if indeed not down to 40 per cent.

We may say, therefore, that — except in pyrite smelting, where there is usually a superabundance of iron — pure ferrous oxide silicate slags are almost unknown, owing to the comparative scarcity of ores high in iron, and because ferrous oxide is too valuable an agent in rendering other silicates fusible to squander it by in using it up to neutralize silica where less choice bases can effect the same purpose just as well, or better.

If circumstances should arise, however, in which it appeared

advisable to make a nearly pure ferrous silicate slag (95 per cent) it is possible — though not advantageous — to go as low as 30 per cent silica; at 35 per cent silica, the slag is very good for the blast-furnace, though corrosive for the reverberatory; at 38 per cent silica, it is at its best for both types of furnace; at 40 per cent, it is still excellent, though already losing a little of its liveliness; at 42 per cent, it is so much less fusible that an addition of limestone is advantageous; at 45 per cent, it is still fusible, although the capacity of the furnaces is now greatly diminished. Above this point, it requires more than ordinary skill to produce it successfully in the blast-furnace, while, in the reverberatory, the temperature must be raised to a point that makes it expensive for fuel and ruinous to the furnace itself.

Lime. — Calcium oxide (CaO) is the typical basic flux, owing to its cheapness, universality, and high replacement-value. The pure calcium silicates are so refractory that they require no consideration from the copper metallurgist. In connection with alumina, lime makes a series of double silicates which form at reasonable temperatures — especially the bisilicate Al_2O_3, 3CaO, $6SiO_2$, which is reported as melting at 1300 deg. C. This compound is more interesting to the smelter of iron ores than to the copper smelter, who usually can depend upon a certain amount of FeO for promoting fusibility; still, in rare instances, even the copper smelter bases his slag upon these acid double silicates of lime (also MgO) and alumina, as in the following slag from the Kochhütte at Mansfeld, the analysis of which was kindly furnished to me by Mr. Anton Eilers:

SiO_2	48.56
Al_2O_3	17.6
CaO	21.81
MgO	3.0
FeO	2.46
MnO	0.32
CuO	0.28
PbO	0.09
ZnO	0.85
NiO, CoO	0.009
K_2O	4.18
Na_2O	0.586
S and C	0.25
	99.995

This slag is a good example of the adaptation of the double silicate of lime and alumina to copper-matte smelting, its fusibility being improved by the presence of small quantities of other bases. In spite of its high percentage of silica, the silicate-degree is quite moderate, there being only 1.44 parts oxygen in the acid to one part in the base, which is thus below a sesqui-silicate. This circumstance results from the high replacement-value of the principal bases present. The high temperature demanded by this slag necessitates the use of a considerable proportion of coke and requires a high ore-column. This brings about a condition approximating that which obtains in the smelting of iron ores, and causes the reduction to metallic iron of a portion of the scanty iron content of the charge. If it were not for the protecting influence of such sulphur as still remains in the charge, it is probable that so much iron would be reduced as to hamper the process seriously.

While a slag based mainly upon the double silicate of lime and alumina may thus be suitable for the copper smelter, where ample fuel is available and where the charge is of such a nature that the high ore-column, slow smelting, and powerful reducing action do not convert too much of the iron into the metallic condition, it would probably not answer in partial pyrite smelting where any large proportion of the heat was to be derived from oxidation of the sulphides. It is not that the partial pyrite process may not produce a sufficiently high temperature at the focus to liquefy this refractory silicate, but rather that the zone of intense heat is so circumscribed that the slag does not receive the vigorous and protracted superheating requisite to maintain it in a thoroughly fluid condition during its passage through hearth and settler. Moreover, we have not yet learned to run partial pyrite furnaces with the high ore-column necessary for the economical preheating of the charge, and for the gradual blending together of the almost infusible constituents.

It must furthermore be remembered that the process of partial pyrite smelting presupposes some reasonable proportion of pyrite in the charge and that, in such cases, the slag would necessarily contain so much ferrous oxide that it would be removed from the class under consideration.

Turning now to the ordinary conditions encountered in practice, we may say, in general terms, that the addition of CaO to

a silicious ferrous silicate is extremely advantageous in several ways. It increases the fusibility of the slag; it lowers its specific gravity; it is cheap, and has a high replacement-value; it lessens the dissolving power of the slag for metal sulphides. These points are understood by all metallurgists and require no discussion.

The important point in actual work is to learn just how far it is possible to proceed in replacing FeO with CaO. Put in a still more familiar form, the question is: how little FeO can I get along with in a highly silicious mixture where limestone is my only available flux? As this is a commercial, rather than a scientific, question, its reply should be based upon actual technical, long-continued results, rather than upon theories or laboratory investigations, or even upon single short furnace-campaigns. Indeed, I have a deeply rooted prejudice against basing such important answers upon any results that did not actually pay dividends, without a thorough personal knowledge of why such an important omission occurred, provided the smelting was a success.

The most extensive and important practical information that we possess regarding these low-iron, high-silica-lime slags comes from the two blast-furnace plants in the Black Hills, South Dakota, which for a series of years smelted the highly silicious non-ferruginous gold ores of that region, using limestone as flux, and collecting the precious metals in a very small proportion of matte — sometimes nearly free from copper. These two illustrations are the more instructive because, in the one case, the limestone was unusually free from other bases while, in the other, it carried a high proportion of MgO, thus affording an opportunity to determine positively whether mere polybasicity implies necessarily increased fusibility. These illustrations are given in detail in the chapter on "Partial Pyrite Smelting" (pp. 300–310), and I will merely recapitulate the results.

In the first place it was found that the dolomitic limestone possessed no especial advantage in comparison with the straight calcium carbonate, excepting that due to the fact that magnesia has a higher replacement-value than lime. So far as fusibility and fuel-consumption are concerned, it seems to make no perceptible difference in the furnace whether a given slag contained 20 per cent CaO and 12 per cent MgO, or whether the entire

32 per cent of earths was represented by CaO alone — the SiO_2 in each case being about 48 per cent.

As the main object of the present inquiry is to determine how far we may go in substituting CaO for FeO, I will proceed at once to the most extreme slags of this nature that were found suitable for regular work. After some years of successful experience, Carpenter, at the Deadwood and Delaware smelter, found that his largest tonnage, least loss of values, and most economical results were obtained with a slag of about the following composition:

SiO_2	51.4	per	cent
FeO	12.2	"	"
CaO and MgO (about 1½ CaO to 1 MgO)	30.0	"	"
Al_2O_3	3.5	"	"
	97.1	"	"

This slag, while having a high flowing-temperature, melted rapidly, was clean, and did not chill or form accretions under proper management. It required about 14 per cent of good coke to melt this charge, which was more than ordinarily difficult from the fact that the rate of concentration was very high — always 20 into one, and occasionally 50 or more into one. This slag lies between the sesquisilicate and bisilicate degree.

It will now be interesting to consider the composition of the slag made at the "National Smelter" in the same district under much the same conditions as in the preceding case, excepting that the limestone used for flux at the National smelter contained practically no magnesia.

During the later years of work at this plant, the slag had, approximately, the following composition:[1]

SiO_2	48.5	per	cent
FeO	11.6	"	"
CaO	33.0	"	"
Al_2O_3	5.5	"	"
	98.6	"	"

This low-iron, high-lime slag would run well for 10 to 12 days, when the furnace would exhibit signs of distress, and it would become necessary to increase the FeO content of the slag to 16 or 18 per cent for two or three days. This would restore normal

[1] See p. 308 for fuller details of this work.

conditions, and the more refractory slag could then be resumed. The very slight preheating of the blast that was sometimes practised at this plant seemed to be distinctly advantageous.

The slags just described are of an extreme nature, and are so high in lime and so low in iron that they melt slowly and demand a considerable proportion of fuel. They are not cited as types that should be followed excepting under the stress of absolute necessity and, even then, should be attained gradually, beginning with at least 20 per cent of FeO, and replacing this base cautiously with CaO, as the workmen become skilled in the management of so difficult a charge. At least 14 per cent of good coke will be required (a certain undetermined portion of which may be replaced by sulphides), the thorough mixing of the charge with avoidance of large lumps is essential, and the height of ore-column and distribution of charge in the furnace must be maintained with regularity and uniformity. No one would make so difficult and precarious a slag unless he was forced to do so; but, on the other hand, no one need hesitate to adopt such a type when circumstances demand it.

It will be noted that, with all of these slags so high in lime and so low in iron, the per cent of silica always remains considerable. This results in part, of course, from the fact that CaO has a higher replacement-value than FeO, so that a slag of the same silicate-degree with CaO for its base will contain a much larger percentage of silica than if its base were FeO (for instance, a lime bisilicate contains 51.72 per cent silica, while a ferrous bisilicate contains only 45.45 per cent silica).

This is, however, not the only reason for the higher proportion of silica that always accompanies commercial slags high in lime. The more basic lime silicates — even when so proportioned that their melting- and flowing-temperatures are not unreasonably high — are treacherous in their behavior, owing to the fact that they have but a narrow margin of safety between complete fluidity and absolute solidification, and thus chill suddenly and block the tuyeres with a tough stony form of accretion that is dangerous to the furnace. The more silicious slags, high in lime, demand fully as much heat as those just cited, but have a wider margin of semi-fluidity between their fluid and solid states, and thus give timely warning of impending freezing.

As this type of slag will scarcely ever be made except where there is an excess of silica, the peculiarity just mentioned is no hardship, and the metallurgist will find that by keeping his silicate-degree between the sesquisilicate and bisilicate, by maintaining his iron oxide in the neighborhood of 15 per cent, and by observing the precautions already enumerated, these slags are entirely feasible, though slow. As one would naturally expect, they are usually low in metal values.

As we increase the proportion of ferrous oxide, the slags become more favorable, and begin to approximate more closely the types adopted at the great copper smelters that are treating silicious raw sulphides with the addition of just that proportion of limestone which they have learned by experience will enable them to smelt in the most economical manner.[1]

As FeO is substituted for CaO the percentage of silica diminishes, even in those rare instances where the same high silicate-degree is maintained as in the high-lime slags that have just been discussed.

The important point for the commercial smelter to note, however, is that *this same high silicate-degree is not maintained* in the class of slags now under consideration and that, in spite of the natural disinclination of the metallurgist to use any undue amount of barren limestone, and in spite of the unfortunate circumstance that his fusible base (FeO) has a decidedly lower replacement-value than lime, so that the more he replaces lime with iron the less silica can he flux, he still finds it advantageous to burn the candle at the other end as well, and actually to lower the silicate-degree of his slag — thus losing ground in both directions. This is the deliberate result reached after years of experience at some of the best smelters in the United States, including the Washoe plant, where the magnitude of operations and the enlightened nature of the management have warranted investigations concerning this point upon a more elaborate and extended scale than any others with which I am familiar. The results of this research work have established beyond a doubt that, with expensive coke, high wages, and generally costly surroundings — and with cheap limestone — it is more profitable to add enough

[1] See the description of the blast-furnace work at the Washoe smelter, page 262, for a fuller discussion of this matter.

barren limestone to reduce the silicate-degree of the slag down to about 1.33 oxygen on the acid side to one on the base side, and smelt rapidly and easily, than it is to attempt a higher silica slag, run slowly, and save the use of a portion of the flux.

The average blast-furnace slag for a long period is about:

SiO_2	39.87
FeO	22.38
MnO	0.6
CaO	26.83
ZnO	1.40
Al_2O_3	6.35
S	0.52
Cu	0.30
	98.25

Any increase in the silica content of this slag above 40 per cent is noted quickly by the men at the furnace, and at 42 per cent there is a tendency to chill at the tuyeres, and a marked diminution in the rapidity of the process.

The same evil results are experienced if any considerable diminution in the proportion of FeO is attempted. This base, during experiments, has been reduced to 17 per cent — with 30 per cent CaO and 41.5 per cent SiO_2 — but was soon followed by chilling at the tuyeres.

It may, then, be said in recapitulation that, apart from FeO — a certain amount of which is absolutely indispensable to promote the fusibility of all ordinary copper slags [1] — CaO is our most universal and advantageous flux for silica, that it may be used freely and without especial precautions until it forms (together with MgO) some 25 per cent of the weight of the resulting slag — provided always that, in reaction-smelting (partial pyrite smelting), the percentage of FeO does not fall below 20, and the SiO_2 does not rise above 40 per cent, or the SiO_2 plus Al_2O_3 above 46 per cent. In straight coke smelting, the ferrous oxide may drop a few per cent lower, and the silica and alumina rise a few per cent without experiencing trouble.

If it is desired to exceed these limits, there need be no hesi-

[1] The sole exception are those rare instances where it is found advantageous to emulate the practice of the iron-smelter and rely upon the double silicate of lime and alumina.

tation in doing it, provided suitable precautions are undertaken and the management is prepared to face the slower smelting and higher fuel-consumption that will result. The SiO_2 may be raised very nearly to 50 per cent, while the SiO_2 and Al_2O_3 together may reach 53 per cent; the FeO dropping to about 15 per cent, and the deficit being made up with CaO.

Magnesia. — Despite the considerable analogy that exists between magnesia and lime, the former earth is the more dangerous and refractory from the standpoint of the metallurgist, although the higher temperatures and more rapid driving of modern furnaces enable us to handle it more freely than in former practice.

Apart from the observation that the substitution of MgO for other ordinary bases is allowed, in general, by a moderate rise in the formation-temperature of the slag, the crucible experiments upon this point do not offer an adequate reason for the decided increase in refractoriness exhibited by slags that contain any considerable proportion of this earth. We must look to the flowing-temperature of such slags for light and, as *mass* is essential before we can determine whether our slag is or is not actually flowable in practice, we turn naturally to full-sized furnace operations as the most reliable guide in determining how far it is practicable to increase the magnesia content.

In the older furnace operations, with lower temperature, slower running, smaller volume of melted product, and generally feebler conditions, we began to notice the chilling effect of MgO by the time the slag contained four per cent of this earth. This statement refers to favorable, fusible, flowable slags, and I know from personal observation that if I added a little magnesian silver ore to the charge, without the knowledge of the furnace foreman, he would begin to complain, by the time the MgO in the slag reached three per cent, that there was something wrong with the charge; and, before it reached five per cent, it was plain to any casual observer that the tuyeres were crusting, the slag-stream losing its luminosity, and the entire operation losing its vitality. This could be corrected, to some extent, by using more coke, but generally at the expense of making iron sows, with their attendant train of evils. If there was much zinc or alumina in the charge, magnesia caused even more serious trouble, and was justly dreaded by every blast-furnace smelter.

In the reverberatory furnace, the results were not so serious from a metallurgical standpoint, because it simply meant keeping the charge an hour or two longer in the furnace until a sufficient temperature was attained to melt it thoroughly; this necessity, however, was a serious matter commercially, owing to the increase in coal and decrease in capacity.

With more energetic furnace conditions, the standard has been so raised that we can now make a satisfactory slag containing 12, and perhaps even 15, per cent of MgO, provided there is not too much Al_2O_3 or too much ZnO present. These substances intensify the refractoriness of the MgO, and tend to the separation of almost infusible spinels ($RO{\cdot}R_2O_3$).

As has been pointed out in the preceding pages, it was found in the partial pyrite smelting, with highly silicious slags, carried on so extensively in South Dakota, that, in making a slag with, say, 48 per cent SiO_2, 13 per cent FeO, 5 per cent Al_2O_3, and 33 per cent CaO, 40 per cent of the CaO could be replaced by MgO without affecting the flowability of the slag, or demanding an increase in fuel. Magnesia, like lime, does not lend itself to the formation of basic slags suitable for copper smelting, but it is evident that, with acid slags, we may use it up to 12 or 15 per cent, provided there is no large amount of alumina or zinc oxide present. If the latter conditions prevail, it is necessary to proceed with caution, as refractory spinel compounds are likely to form.

Baryta.—As this substance, when present in ores, is usually in the form of barite ($BaSO_4$), we have first to decompose this very stable salt before the BaO is in condition to be slagged by the silica. This matter has been discussed on page 55, and I will simply state that, in an oxidizing atmosphere, barite is decomposed, and barium oxide set free to unite with silica. As an oxidizing atmosphere is an essential feature of this reaction, and as barite is not decomposed in the moderate temperature of the roasting furnace, it was impossible to smelt heavy-spar ores in the coke-burning blast-furnace without producing an excessive amount of matte, as well as a BaS product which contained values, and which would not separate properly from either the matte or the slag. The reverberatory furnace decomposed barite more successfully, but it was not until the introduction of partial

pyrite smelting that we were enabled to effect this purpose in a cheap, rapid, and efficient manner, and make slags containing 25 per cent BaO, and more, while maintaining a satisfactory degree of concentration. Both the Tyee mine in British Columbia and the Kosaka mine in Japan execute this kind of work with entire success, and it is one of the important metallurgical advances of the times.

Owing to the extraordinarily high equivalent-value of BaO (153), it would be a peculiarly unsatisfactory base to use as a flux for SiO_2, but where it is required to slag barium oxide already present, it may probably be run up to 30 per cent without serious difficulty excepting such as results from the high specific gravity of such a slag.

Zinc Oxide.—Zinc blende is regarded by the metallurgist as a disagreeable substance to be got rid of in any way possible rather than as a potential base for excess silica. In ordinary coke blast-furnace smelting and, to some extent, even in the reverberatory furnace, much of such ZnS as is in the furnace-mixture is not oxidized at all, some of it being dissolved in the matte, some of it entering the slag mechanically, and some of it volatilizing at a very high temperature.

At present the study of zinc must be confined to the behavior of its oxide in connection with the formation of slags. It has long been known by practical smelters that, if it is desired to slag any considerable quantity of zinc oxide, the iron should be high and the lime comparatively low. Vogt shows that this necessity arises from the fact that ZnO tends to enter the olivine group of minerals, which require high iron and low silica for their formation.

In modern times the development of partial pyrite smelting enables us to handle ores higher in blende than would formerly have been considered practicable. The powerful oxidation reigning in the focus of the furnace decomposes thoroughly the refractory blende, and the high temperature promotes the slagging of such zinc as escapes volatilization, and superheats the slag sufficiently to enable it to flow satisfactorily even when containing eight per cent, and more, of zinc oxide.

Too much alumina and magnesia, and too little ferrous oxide, are the conditions most inimical to the production of a slag high

in zinc oxide: the former two substances because they tend toward the separation of refractory zinc spinels, and the latter for the reasons already specified.

Alumina. — This substance has provoked more discussion — accompanied by less results — than any of the other bases (or acids) that are found in ordinary slags.

In my own metallurgical contributions I have been accustomed to class it among the bases — not from any profound conviction upon the subject, but because it seemed to me more convenient so long as I could see no sufficient reason to regard it as an acid. This habit is confirmed by the investigations of Vogt, who finds that if the composition of the slag is such that mineral compounds will form which carry alumina as a base (namely, in the ortho-silicates, melilite, gehlenite, or anorthite, and in the bisilicates, augite) the alumina will enter the slag accordingly; while, if this is not the case, the alumina may act as an acid, forming refractory spinels with ZnO and MgO.

This view of the case seems to me to be the most reasonable yet advanced, and accounts also for the extraordinary behavior of earthy high-silica slags carrying a moderate amount of alumina, and which will — for instance — be perfectly satisfactory at 45 per cent silica, or at 48 per cent silica, but midway between these two extremes will develop a refractoriness that threatens to freeze the furnace. It is possible that, in the latter cases, the proportion between silica and bases is not suitable for the formation of those minerals which take up alumina as a base, and that a considerable amount of comparatively infusible spinel is formed suddenly, which is too much for the already refractory slag to hold in solution.

Under suitable conditions, and after careful experimentation to determine just how his silica and other bases must be proportioned to induce his alumina to enter the slag as a fusible silicate rather than a refractory aluminate, the skilled metallurgist may be able to produce commercial slags carrying a percentage of Al_2O_3 higher than I dare to cite; but, for ordinary work, 10 per cent of this earth will be found to cause sufficient viscosity and general chilling tendency to occupy the earnest attention of the furnace-men.

CHAPTER XVIII

MISCELLANEOUS

THE details of furnace construction are almost endless and cannot be acquired from books. Actual long-continued contact with smelting operations, combined with a certain — even if slight — measure of responsibility for results, is the only means of obtaining a sound education in this branch of the art. Without going into details, however, except for the sake of illustration, it is possible to deduce from experience certain general principles that may serve as fixed points to which the younger man may anchor his more specific facts, as they gradually accumulate.

The important question of foundations seldom receives sufficient attention from the beginner. The evil effects of an insufficient foundation become apparent so quickly and so seriously in a chimney, or other tall structure, that there is not much danger of its being slighted in such extreme cases; but a reverberatory furnace, or a water-jacket blast-furnace, or a converter-stand, or a rotary blower, is occasionally treated more carelessly, and is sure to cause an amount of expense and trouble that would pay several times over for a proper foundation, besides bringing serious discredit upon the builder.

Mere depth in the ground stands in no particular relation to a solid foundation. The object of depth is to get solid ground, and away from frost and flood, and when this purpose is accomplished, there is nothing gained by going deeper. If solid ground cannot be reached at any reasonable depth, a large hole should be excavated and filled with a solid block of concrete, or with a monolith of slag poured in from a neighboring furnace, the object being to lessen the danger of any one portion sinking, by spreading the weight equally over a large area.

Mortars are sometimes a perplexing subject to the person inexperienced in metallurgical construction, but need never remain so after a moment's reflection. The object of mortars, at ordinary temperatures, is two-fold: to fill the openings, and to hold the brick

or stone together. This latter power depends upon the formation of certain compounds in the mortar which are destroyed by high heat. Consequently, this sort of mortar loses its cohesive value entirely when exposed to heat, and at once shows it own limitations.

The three ordinary varieties of mortar familiar to the smelter are lime mortar (often with an addition of cement to improve its qualities), fire-clay mortar, common clay mortar (often termed *mud*).

The lime mortar is used where strength is desired, and where no considerable heat prevails. The fire-clay mortar is used, naturally, to withstand high temperatures, and in connection with equally refractory brick or stone. The common clay mortar is used in situations where it is too hot for the strong lime mortar, and not hot enough for the more costly fire-clay. Consequently its main employment is with red brick.

Besides these three ordinary mortars, there are many special kinds for special situations — usually where great heat prevails. They are generally made out of the same material as that of which the particular variety of brick is formed which demands their employment, so that we shall feel sure that we are not diminishing the valuable refractory quality of the brick by laying it with mortar that will not stand as much heat. For instance: silica brick, employed so commonly for the interior of reverberatory smelting furnaces, are not laid in fire-clay. If this were done, the very object for which we employ the costly fragile silica brick would be defeated. We use them because they are a great deal more refractory to heat than ordinary fire-brick, and if we should lay them in fire-clay mortar, this would begin to melt at the joints, and would shorten the life of the wall or arch, or even destroy it completely. Instead of fire-clay, we use some of the very silica of which the brick are composed. The same is true of magnesite and bauxite brick, and of various others.

Joints in brickwork that is exposed to high temperatures are, at the best, a source of weakness, and a vantage-point for the fluxing action of the coal ash or ore dust. Hence we eliminate them, so far as practicable, by employing the least possible amount of mortar. Indeed, the experienced furnace mason, in critical situations, does not employ any mortar at all in the ordinary acceptation of the term. Fire-brick of all kinds are pressed and

burned with great care, so that they are generally smooth and unwarped. Consequently, they would lay into a solid and tolerably tight wall, even if laid dry. In order to fill the cracks as nearly air-tight as possible, the furnace mason's mortar is about the consistency of thick cream, and the contact surfaces of the brick are simply dipped into this dope, and the brick tapped firmly into position until it takes a solid bearing all around.

The refractory materials of importance to the copper smelter may be classified as follows:[1]

I. Silicious refractories.
  (*a*) Fire-clay, aluminous fire-clay, silicious fire-clays.
  (*b*) Dinas and silica brick.
  (*c*) Gannister — quartz furnace-sand.

II. Basic refractories.
  (*a*) Magnesite and magnesia.
  (*b*) Bauxite.

III. Neutral refractories.
  (*a*) Chromite.
  (*b*) Graphite.

Most of these refractories may be obtained either in crushed or ground condition, as standard brick, or as special shapes, designed for some particular use.

I. *Silicious Refractories.* (*a*) Fire-Clay and Fire-Clay Brick. — This is, of course, the most important division in point of quantity. The essential constituents of all fire-clays are alumina and silica, which usually comprise 92 to 98 per cent of its entire weight, the remainder consisting of impurities which have a decided effect in lowering the refractory qualities of the clay. The most common of these are ferric oxide, lime, magnesia, and the alkalies. The silica content of a clay is usually present in three different conditions: as free silica, or quartz sand; in combination with alumina to form the plastic mineral kaolinite (a hydrated silicate of alumina), the most important constituent of the clay, and the one which makes it plastic; in combination with some of the impurities just enumerated (unimportant).

The qualities of each brand of fire-brick are well known, and

[1] In this résumé of refractories, I have followed, in part, the classification and descriptions used by C. H. Fulton in "Principles of Metallurgy," Chapter X.

the metallurgist will make his selection in conformity with the conditions of his process.

(*b*) Dinas Brick, Silica Brick. — Dinas take their name from a town in Wales where a bed of very pure sandstone exists which, after calcining and quenching with water to effect disintegration, is mixed with a very small proportion of milk of lime, moulded with great care, and burned in kilns at a temperature sufficient to cause a formation of lime silicate which acts as a cement to bind the sand grains together, though in a somewhat precarious fashion. Genuine Dinas brick contain over 96 per cent silica.

Silica brick, in which the binding material for the quartz grains is usually clay. The term is loosely applied, and does not necessarily guarantee a high degree of refractoriness. Where this is desired, the silica content should not fall below 95 per cent. Such brick are largely made in this country, and many of the brands are entirely satisfactory.

The term "gannister," which was originally applied to a dark clayey sandstone used in England for converter-linings in the steel industry, is now in more general use, signifying any silicious rock which contains clay enough to hold the grains together, and is comparatively free from fluxing bases. At present, in the copper industry, it is often made artificially from crushed sandstone or quartzite and binding-clay, although the modern tendency is to replace the barren material with quartzose ores.

Furnace-sand, for the bottoms, or hearths, of copper reverberatory smelting furnaces is still used extensively, although it is a mooted question whether brick bottoms may not be substituted with advantage, as has been done completely at some of the large English copper works, both for ore furnaces and refining furnaces. Numerous analyses of successful bottom-sands are available, but are, to my mind, rather misleading than otherwise, as the suitability of a given sand seems to stand in no definite relation to its chemical composition. In a general way it may be said that a sand containing less than 92 per cent silica would seldom stand the wear of a smelting-hearth, and that 95 per cent of silica is not an uncommon standard. The agglomeration of the sand particles into a safe and solid bottom is not effected so much by any fusion of the surfaces of the particles themselves, as by the soaking and

cementing action of the ore or metal that is melted upon the hearth.[1]

II. *Basic Refractories.* (*a*) Magnesite and Magnesia. — Magnesite is the natural mineral from which magnesia is obtained for the purposes now under consideration. It is imported, mostly, from Greece and Styria, is calcined in kilns, and after mixing with about 10 per cent of incompletely burnt magnesite, is compressed into brick and burned. They are exceedingly refractory, withstanding 1800 to 1900 deg. C., and melting at about 2000 deg. C.

Their disadvantageous qualities are: inability to stand sudden changes of temperature without flaking and spalling, greater expansion and conductivity for heat than ordinary fire-brick, tendency to soften (not to melt) at high temperatures, so that they will not bear a heavy load, general mechanical weakness. Nevertheless, magnesia brick form a class of material of the greatest value in metallurgy, owing to their refractoriness, and their ability to withstand the corrosive action of metal oxides to a far greater degree than the materials commonly used. Magnesite brick are employed largely in lead refining and cupeling furnaces, in the lining-walls of copper refining furnaces, in the "tests" of cupeling furnaces, in crucibles and settlers for copper furnaces, and in many situations peculiar to the iron and steel industries. Thus far they have proved the most suitable material for replacing the acid lining in copper converters.

Magnesite brick are either laid dry, or with finely-ground magnesia mixed with only enough tar to make it plastic, and kept hot in pans, so that the joints may be as thin as possible.

(*b*) Bauxite. — This is a hydrated oxide of alumina and is usually quite impure, containing much ferric oxide, with some silica and titanic acid. It is the main ore of aluminum, but its valuable refractory qualities are causing its increased employment for metallurgical construction. The Arkansas bauxite is washed at the mines to remove free silica, and is then calcined at a temperature of 1400 deg. C. The calcined material is then bonded with a very small proportion of fire-clay, lime, or sodium silicate, moulded into the desired shape, dried, and burned at a high temperature. These brick are peculiarly hard and tough,

[1] See chapter on "Reverberatory Furnaces" for a more extended discussion of this subject.

and an ordinary specimen will stand a crushing test of 10,000 lb. per sq. inch. This is an exceedingly strong, tough, and refractory material, and will be valued highly by those who use it in appropriate situations. For lining the fire-boxes of reverberatory furnaces, especially where the ash is neutral or basic, they last much better than silica brick, and in the flue of such furnaces, they have been found advantageous.

III. *Neutral Refractories.* (*a*) Chromite. — This is a natural chromate of iron containing, when pure, Cr_2O_3, 68 per cent; FeO, 32 per cent. Commercial chromite, as used in this country, comes mostly from New Caledonia, having much MgO in place of FeO, and one-fifth or more of its Cr_2 replaced by Al_2O_3. This is an extraordinary substance, resisting to a remarkable degree the chemical action of both basic and acid slags, withstanding mechanical wear and fluctuations of temperature better than any other refractory material, and resisting the highest temperatures known to the copper industry. The main obstacle to its more extensive use is its high cost, which is five to six times as great as that of good fire-brick.

(*b*) Graphite. — This is about the most refractory substance known, withstanding a temperature of over 3500 deg. C. As it is destroyed rapidly when exposed to oxidizing conditions, it is not suitable for extended use in the metallurgy of copper. Its most familiar use is for the manufacture of crucibles where the graphite, or plumbago, is mixed with at least an equal weight of fire-clay. Bricks of this type are also made from graphite and tar.

The general arrangement of plant and buildings must depend, to a considerable extent, upon the source of the ore-supply as well as upon the configuration of the available site.

There are two principal types of arrangement, each of which has its advocates; but the general view of most constructing metallurgists of the present day is that, under ordinary conditions, a compromise between the two extremes secures a more moderate initial cost combined with reasonable economy of operation.

The hillside, or terrace, construction possesses the great advantage of delivering by gravity the ores and products from one operation to the next, whereby economy of time, space, power, and installation is gained. On the other hand, the cost of grading is large, and the plant is apt to be crowded and to get into an

awkward and costly condition when it outgrows the limits of its original slope.

The horizontal location, while offering cheap grading and unlimited space for extension, involves a permanent running expense for the several successive lifts of material which will be required even in the simplest process.

Where the configuration of the ground is at all favorable, our most experienced and successful constructing metallurgists endeavor to combine the two systems in such a manner as to enjoy a considerable proportion of the advantages of each plan without pushing it so far as to suffer unduly from its drawbacks.

A clear analysis of the operations through which the ores, fluxes, and fuels must pass from the moment that they enter the plant until they leave it as slag and matte must, of course, precede every attempt at designing a new smelter. In order to shorten the discussion, I will assume a situation of extreme simplicity where sulphide copper ores, using limestone as flux, are to be subjected to partial pyrite smelting in the blast-furnace. The matte will pass to converters, and the resulting copper will undergo a partial refining in an anode furnace. This simple straightforward chain of operations will illustrate the method of studying the subject more clearly than if it were encumbered with the complications, so often present, of concentrating-mill, roasting-plant, and reverberatory smelting furnaces.

Leaving aside, for the moment, any question of horizontal movement, it is evident that the materials must be delivered at several different levels, regardless of whether they attain these various horizons by mere gravity — as in the hillside plant — or by undergoing a series of liftings — as in the horizontal plant. The most important of these levels are:

(*a*) The unloading level, where cars or wagons may discharge into storage bins situated on a lower plane. Even in horizontal plants this level is established artificially by a long trestle constructed on a grade that will permit cars to run above the bins.

(*b*) The storage level, upon which are arranged the bins for receiving the raw materials and between-products that are to go to the blast-furnace. As the discharge of these bins — whether onto conveyors or onto charging-cars — is some distance above the base of the bins, we may assume that the storage-level and the charging-level are identical, making due provision for the vertical

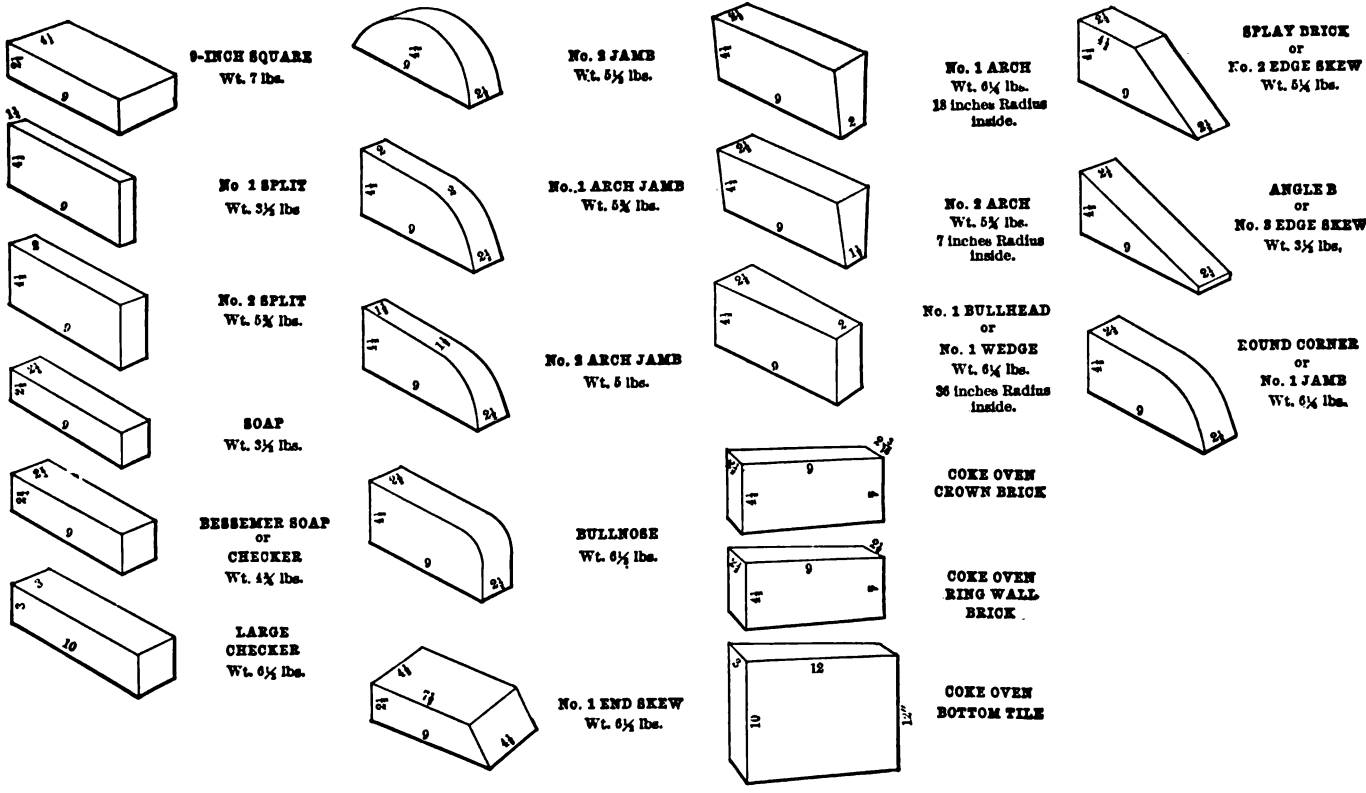


FIG. 116. — Ordinary shapes of refractory brick

distance required between the discharge-spout of the bin and the track upon which the charging-car runs.

(*c*) The furnace-floor level upon which the main work of the blast-furnaces is executed.

(*d*) The slag-dump level, by which I mean the horizon upon which the slag-dump rests. The vertical distance between (*c*) and (*d*), minus the slight deduction for grade, represents the possible depth of the future dump.

Amongst these fundamental horizons I have not mentioned the converter-floor, because this process, while an indispensable link in the present chain of operations, is only a side issue as regards the general downward course of materials from their original delivery point to the slag-dump.

It must be recollected that it is the weight and volume of materials, and not their intrinsic value, that concerns us when providing for their handling, and that the portion that passes from the blast-furnace to the converter is small in weight, and particularly small in volume. Furthermore, the products of the converter — bar-copper, slag, and cleanings — do not return to the general channel, but pursue an entirely separate course — the former to the blister furnace, the latter upward again to the storage-bin level above the blast-furnaces.

Having established clearly the fundamental horizons of a prospective plant of the nature indicated, we should next consider the horizontal movements which the various materials must make in passing from one to another of the operations that constitute our process. This is a much more important consideration at the present era of large enterprises than it was a generation ago before the introduction of belt-conveyors and similar appliances. Formerly, we used almost exclusively some form of vertical elevator, or a short inclined plane. This was reasonably satisfactory so far as overcoming the vertical movement was concerned, but did not provide adequately for the horizontal distance. Thus, for instance: the transportation of the converter slag from the converter shed to the storage bins above the row of blast-furnaces required that, after loading it into cars at the converters, the car must travel a considerable distance horizontally until it reached the elevator; here it was raised to the level of the track running above the storage bins, and had to make a second horizontal movement to reach the point where it could dump its

contents into the proper bin. With the enormous tonnage now handled at smelting plants, this system is cumbersome, expensive, and inadequate.

The remarkable efficiency attained by conveyor-belts enables us now to combine the three horizontal and vertical movements in a single inclined one, and furnishes us also with a system so elastic that it will discharge automatically at any desired point and is, practically, unaffected by any variations in load between zero and maximum. This admirable system of transportation has modified profoundly our older ideas as to arrangement of plant, and has rendered us more independent of the terraced formation, and much less limited as to area. On the other hand it is well not to abuse our newly-acquired luxury, and to recollect that, where conditions are suitable, gravity is cheaper and much more permanent than even the most perfect mechanical appliance. Consequently, when a favorable site can be found, the acme of simplicity and economy may best be attained by uniting the two systems.[1]

Assuming that, in general, it costs about the same to handle material whether it is barren or valuable, it is evident that our most anxious thought, as well as our most liberal expenditure, should be devoted to the low-grade material, simply because there are more tons of it to handle.

The maximum amount of material to handle is at the point of original delivery where ores, fluxes, and fuel are received in their raw condition and, consequently, in their heaviest and bulkiest form. For this reason it is well to call in the aid of gravity at the delivery point, and to establish a thoroughly adequate and judicious system of bins for the reception of all raw

[1] The detailed study of this question can best be undertaken at some of the large modern plants, choosing only such as have been erected so recently that their general features are not hampered by the necessity of conforming to some older nucleus, and spending sufficient time and study to learn their weak points rather than their good ones.

The first day's examination of a reasonably good modern plant usually brings out only its advantageous features. We admire the many points of excellence that are new to us, and obtain a general impression that everything is admirable. When familiarity has dulled this initial impression, and we have conversed at length with the foremen of the various departments, the weaknesses and imperfections of the arrangement begin to appear and, from this point, our observations are likely to possess greater practical value than during the early portion of the visit.

materials and for their delivery, as needed, to the furnaces below, and to construct above these bins lines of rails on a grade of sufficient moderation to obviate constant delay and annoyance in running the ore trains up to this level.

Under ordinary circumstances a steam locomotive, with its weight concentrated upon the drivers, is most satisfactory for this preliminary work. Where cars of smaller size are required within the plant itself, steam, electricity, and compressed air are the customary agents employed for power, compressed air being the safest and most advantageous, but being so expensive to install

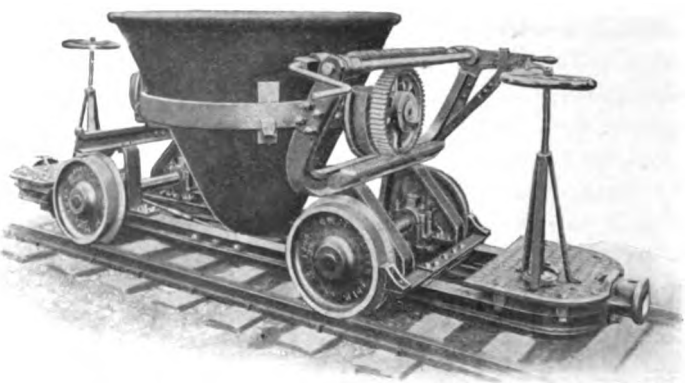


FIG. 117. — Screw-dump slag-pot

that it demands a production of at least five million pounds copper per month.

While electrically-operated motors are satisfactory for handling charging-trains, or even for conveying liquid matte, they are not so suitable for the slag-dump, steam being preferable in this situation. Where water-granulation of the slag is not feasible, the ordinary screw-dump slag-car, operated in trains by steam power, is the most satisfactory method of transporting the slag.

The liquid matte is conveyed from smelting furnace to converter in ladles mounted upon trucks similar to the slag-pot just illustrated, or by means of the traveling electric crane. At works of any considerable size it is seldom practicable to tap the matte into the converter direct from the smelting furnace; but, as at

Anaconda, it may be convenient to pour it from the ladles into a launder above, and back of, the converters, and allow it to run by gravity into the vessels which receive the fresh charge while still remaining in their normal vertical position. This avoids the confusing and dangerous practice of bringing large quantities of liquid matte, suspended in the air, the entire length of the converter shed and above the heads of the workmen, and diminishes greatly the heat, smoke, and danger of the process, and also relieves the electric crane of much duty.

This traveling electric crane is one of the most important adjuncts of the modern converter plant, and is too distinctly a specialized machine to be discussed in these brief remarks. Both strain and danger are increased greatly with augmented speed of bridge-travel, and I doubt if there is any gain in having the crane geared to make more than 140 ft. per minute with the short runs commonly in use.

An unlined cast-steel ladle with detachable bale, and capacity sufficient to hold a full converter-charge of matte, is the simplest and most suitable vessel for matte of fair grade; lower grade material will require a lined ladle.

If the converter slag is to be re-melted in reverberatory furnaces, it is poured into the latter from the ladle through a hole in the roof fitted with a projecting cast-iron launder. If it is to go to the blast-furnace, it is nearly always fed to the latter in solid form, if only to obtain the advantage of its loosening effect upon the charge. As each ton of pig copper turned out involves the production of about two tons of converter slag, it is worth while to expend considerable care and money in perfecting the details of its removal and transportation.

There are various mechanical appliances for this purpose, most of them consisting, in the main, of a wrought-iron or steel frame made of I-beams, mounted on roller-bearing wheels, and moving like a "merry-go-round."

The accompanying illustrations from *Engineering and Mining Journal* illustrate a small type of this apparatus designed by Kilker for handling matte. The same principle is applied on a larger scale, and with automatic dumping, to slag from the converters, and the moulds may discharge direct into an endless conveyor which transports the slag to the bins above the blast-furnaces.

The ore-bedding system, by which a uniform furnace-mixture is obtained by applying ingenious and elaborate mechanical devices to the old ore-bedding system so long practised on a moderate scale by the lead-silver smelters, has become developed into such a complete and far-reaching chain of operations that I am compelled, reluctantly, to forego any attempt at describing it in this work.[1]

The use of heated wind for copper blast-furnaces has already been discussed at considerable length, and it has been shown that experience teaches that, in true pyrite smelting when conducted under typical conditions, with powerful blast, high ore-column, and vigorous oxidizing-effect within the furnace, heated blast tends to lower the ratio of concentration and to increase the acidity of the slag. As both of these results are inimical to the operation of true pyrite smelting, the employment of a heated blast in this process cannot be recommended unless for exceptional conditions.

In the now somewhat rare process of pure coke-smelting in a reducing atmosphere, the employment of heated wind might possibly be advantageous, especially under such conditions as those obtaining at Mansfeld where a non-ferruginous slag, deep ore-column, and high fuel-ratio approximate the situation of the iron-ore furnace. Ordinarily, however, the fuel required to heat the blast can be utilized more advantageously within the furnace than in a separate appliance.

The typical application of heated blast in copper smelting comes in those instances of partial pyrite fusion where sulphides are scant, slag silicious and low in iron, and where it is difficult to obtain sufficient heat to melt the refractory charge without using so much coke as to spoil the ratio of concentration, and rob the slag of its scanty iron. There may be other cases in which a hot blast is useful but, under the conditions just enumerated, it becomes indispensable for the best results.

The most obvious means for the moderate preheating of the blast employed in copper smelting is the waste heat of the gases

[1] Those interested in this matter should study the practical working of this remarkable system at Cananea, Copper Queen, and other smelters. Useful descriptions of the system may be found in *Mineral Industry*, Vol. XV, p. 249; *Mines and Minerals*, Sept., 1909, and *Eng. and Mng. Journal* of Aug. 11, 1906.

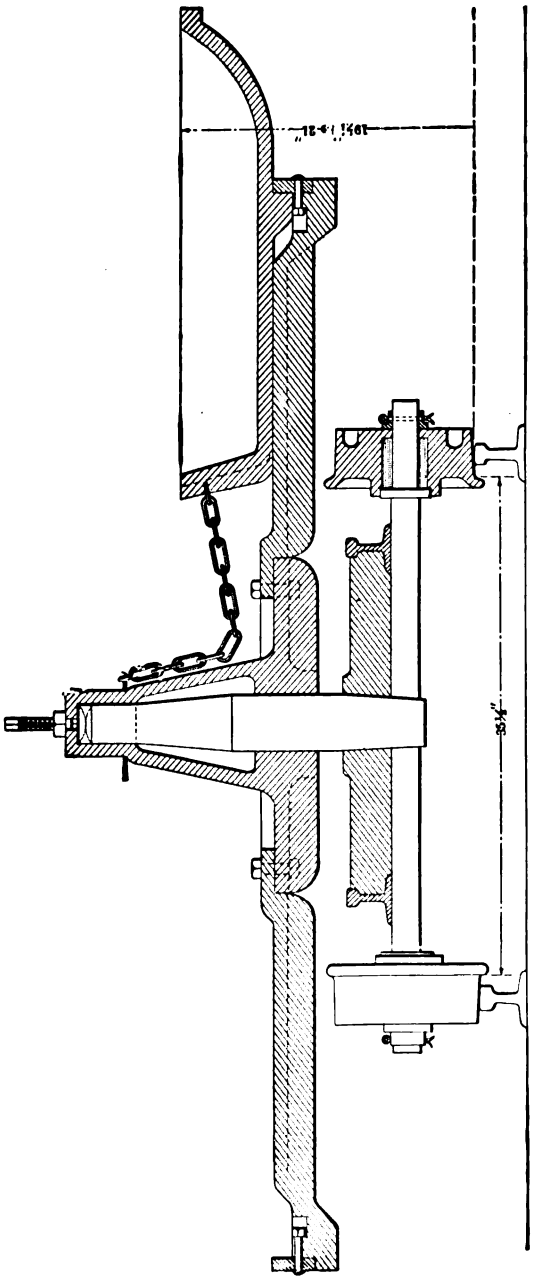


Fig. 118. — Tapping car; sectional elevation

escaping from above, or from the liquid slag below. The greatest difficulty encountered in heating air by either of these methods is that, owing to its low conductivity, it is necessary to maintain the air which is to be heated in contact, for some appreciable length of time, with the surface from which the heat is to be derived. This is effected in some cases by enclosing the tunnel-head of the blast-furnace with air-jackets through which the blast passes on its way to the tuyeres.

Kiddie's system of utilizing the waste heat of the furnace gases for preheating the blast is based upon an extensive and gradually increasing exposure of the air to be heated to the waste gases of the furnace, exposing it first to the heat of the dust-chamber farthest from the flue and, thereafter, to the greater heat of the direct gases close to the tunnel-head. His system contemplates a long exposure in thin steel pipes to the moderate temperatures of waste gases rather than the customary short exposure in cast-iron pipes to the high temperature evolved by the direct burning of coal for the purpose. At the Tyee smelter, where this system is in use, the temperature of the blast averages 175 deg. C., that of the waste gases being 260 deg. C.[1]

The large amount of heat — sensible and latent — contained in the slag issuing from an ore furnace has attracted the attention of inventors for the past 100 years or more. I have personally witnessed many attempts to utilize some of this waste heat for preheating the blast as well as for producing steam, and, while certain beneficial results have been obtained, the general cost, inconvenience, and mechanical difficulties have always outweighed the advantages experienced. The general principle that has been employed in forcing the slag to give up a portion of its heat to the blast consists in moving conveyor-pans — or similar vessels — containing the hot slag through a closed chamber traversed by the blast. The cooled slag passes to the dump at the far end of the chamber, while the heated wind leaves the chamber at the other end.

As both calculation and experiment show that 2000 lb. of

[1] For the information of those who have not had personal experience in the use of slightly heated wind in difficult cases of partial pyrite smelting I will say that even the most trifling warming of the blast is attended with marked benefit in typical instances, while 150 or 200 deg. C. may divest the smelting of much of its difficulty.

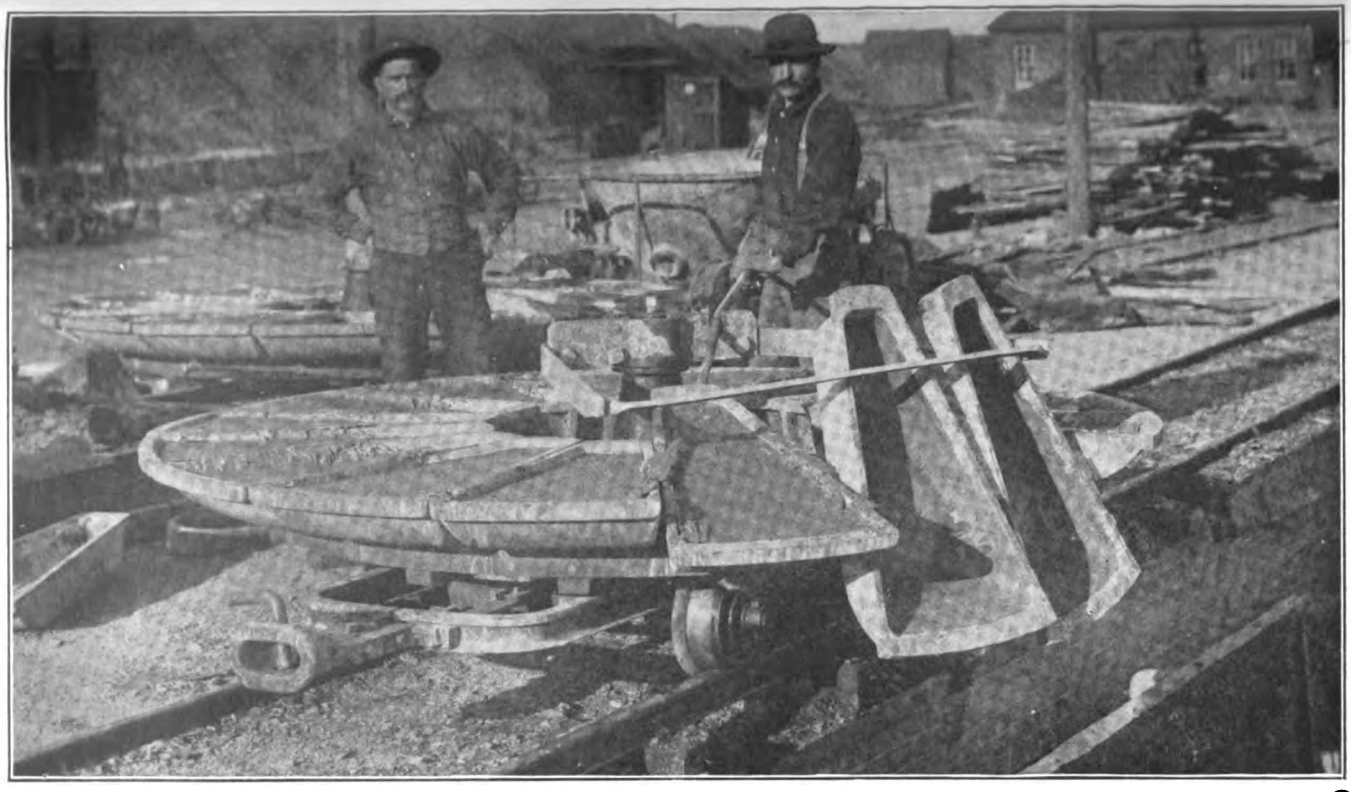


FIG. 119. — Tapping car

ordinary slag leaving the furnace at a temperature of 1350 deg. C., will produce an effect in heating air moderately equal to 100 lb. of good bituminous coal (this is probably an extremely conservative estimate), worth perhaps $0.30, it follows that a furnace producing 300 tons of slag daily would be equal to $90 worth of coal in preheating the blast, provided the mechanical details of the installation could be worked out in a satisfactory manner.

Both air and slag are easy substances to handle and would have long since been made mutually advantageous were it not for the fact that they are both poor conductors of heat. Consequently, the exterior surface of even a thin cake of slag may be comparatively cold while most of its heat is locked up in its red-hot center; on the other hand, the exterior of an air current which is in contact with hot slag may have attained a satisfactory temperature while the remainder of it is still comparatively cold. An intimate mixture of the air and the liquid slag seems essential to any satisfactory transference of the waste heat of the latter substance.[1]

The ordinary apparatus employed for heating the blast with the aid of extraneous fuel is the familiar U-pipe stove. This device is, on the whole, the most universally suitable means for the moderate requirements of the copper smelter who intends to heat his blast by the aid of coal, wood, or similar extraneous fuel,

[1] Vautin's device for utilizing the heat of the slag in the production of steam for power is based upon the invention of the low pressure steam-turbine (one-pound gauge), and offers possibilities which will attract the interest of metallurgists. The inventor states that three fundamental conditions are essential to the utilization of slag for the production of steam for power; these are:

(*a*) The use of the steam at or about atmospheric pressure. This condition is fulfilled by the employment of the modern low pressure steam-turbine.

(*b*) The continuous admission to, and removal of the slag from, the steam generator. This is accomplished by means of a specially designed trapped slag-inlet for the admission of the molten slag, and of a suitably trapped elevator for the removal of the granulated slag from the steam generator.

(*c*) The prevention of any escape of steam from, or of the admission of air into, the generator at the point of slag-admission and removal. This is effected by means of a trapped slag-inlet which admits the slag while shutting out air and, at the same time, preventing the escape of steam. One horse-power per ton of slag per day of 24 hours is claimed for this device.

and who does not find it advantageous to employ his waste gases for this purpose, as in iron smelting.

Serious complaints have been made by some metallurgists regarding the efficiency, duration, and general utility of this device, while others find it entirely satisfactory. As a matter of fact, the apparatus itself is sufficiently good, provided certain fundamental conditions are complied with.

A U-pipe stove must be properly designed, and must be constructed of material suited to withstand the severe work demanded of it. It must be erected upon a solid foundation with care and skill, and in such a manner that the pipes shall be protected from direct impingement of the flame issuing from the fire-box.

As air is heated almost solely by direct contact with a heated surface, and not by radiation, a large area of such heated surface must be provided if a fair degree of efficiency is expected. The stove must not be forced beyond its powers by excessive firing, an increase of capacity being obtained by augmenting the area of moderately heated surface over which the air passes, rather than by heating a limited surface to an excessive temperature.

If these conditions are observed, a properly constructed U-pipe stove, maintained at a uniform temperature, will demand few repairs, and will give a good efficiency for a long period of time.

These stoves are usually built in sections containing, say, 28 U-pipes in four rows of seven each. The pipes themselves are of cast iron from $\frac{3}{4}$ in. to 1 in. thick, 6 in. to 10 in. inside diameter, and 9 ft. to 18 ft. in length (18 ft. to 36 ft. for the complete U). The pipes are connected above the heating-chamber by flange joints provided with asbestos gaskets. Under proper supervision there need be no leakage of air, and a defective pipe can be replaced within an hour by shutting off the blast and deadening the fire.

The roof of the heating chamber is formed by fire-tiles resting upon flanges attached to the U-pipes below their upper bend. This roof, through which project the elbows of the U-pipes, is covered with a thick layer of ashes to prevent radiation of heat.

Figure 120 illustrates one section of a typical U-pipe stove, a

portion of the side wall being removed to show the pipes suspended in the heating chamber.

Figure 121 shows how the interior heating surface of the pipes is doubled by longitudinal ribs.

Experience shows that, when the foregoing precautions are observed, cast-iron is a more suitable material for U-pipes than either wrought iron or steel.

Depreciation of Plant demands the setting aside annually of a specific sum sufficient to cover the original cost in a certain number of years; hence it is important to know what length of time may be allowed for the life of a plant, in the light of experience.

A modern smelting plant, maintained constantly at a high degree of efficiency, has a much longer life than the older smelting works. Analysis of a considerable number of cases leads me to believe that a period of about 17 years may be taken as a conservative estimate for the efficient life of a modern smelting plant, and that, consequently, an annual amortization charge of six per cent on the original cost would be suitable.

The dismantling of an abandoned plant yields such pitiful results, in proportion to first cost, that even the most conservative estimate will usually be too high. E. P. Mathewson[1] gives the actual salvage recovered upon dismantling a great copper concentrating and smelting plant (no doubt the plant used by the Anaconda Copper Mining Co. previous to the erection of the Washoe smelter). Forty-four per cent of the equipment would yield no salvage; 2 per cent would yield 2 per cent of its original cost; 11 per cent would yield 5 per cent of its original cost; 37 per cent would yield 10 per cent of its original cost; 2 per cent would yield 15 per cent of its original cost; 2 per cent would yield 20 per cent of its original cost; 2 per cent would yield 25 per cent of its original cost. The 100 per cent would yield 5.49 per cent of the original cost.

Neither the size nor the intention of this work permits of any extended review of the costs of producing copper. Indeed, this matter is influenced so profoundly by the character and situation of mine and smelter that mere figures, unaccompanied by detailed descriptions, can possess but little value. Still, a few typical

[1] *Engineering and Mining Journal*, Nov. 10, 1906, p. 888.

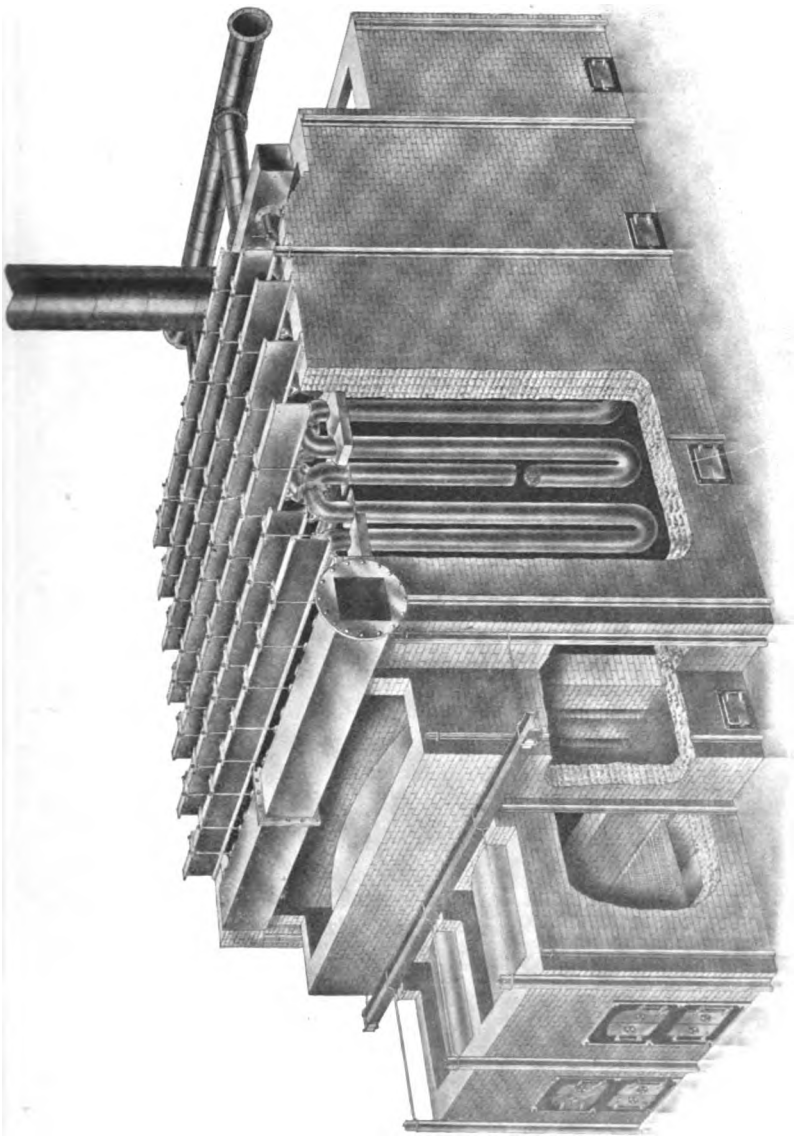


FIG. 120. — U-pipe stove for heating blast

results from large, well-managed plants may serve, to some extent, as guides.[1]

The following table gives the approximate costs of mining and smelting ore at some of the prominent copper mines of Butte, Montana:

COSTS AT MONTANA COPPER MINES

ANACONDA COPPER COMPANY

(Transportation to Anaconda 26 miles)

	Tons	Mining Per Ton	Freight to Smelter Per Ton	Reduction Per Ton	Refining-Marketing Per Ton	Total Cost Per Ton
1903 ...	1,392,835	$3.49	$0.15	$3.39	$2.30	$9.33
1904 ...	983,001	3.73	0.15	3.82	1.96	9.66
1905 ...	1,473,644	3.56	0.15	3.00	1.11	7.82
1906 ...	1,521,310	3.63	0.15	2.27	1.08	7.13
1907 ...	1,401,948	4.47	0.16	2.52	0.93	8.08

BOSTON AND MONTANA

(Transportation to Great Falls)

1903 ...	907,227	$2.61	$1.00	$3.05	$2.90	$9.54
1904 ...	988,866	2.89	1.00	2.53	1.81	8.23
1905 ...	1,138,307	2.91	1.00	2.21	1.69	7.81
1906 ...	1,209,805	3.45	0.93	2.45	0.90	7.73
1907 ...	1,156,785	3.93	0.76	2.67	0.92	8.28

BUTTE AND BOSTON

(Transportation to Anaconda)

1903 ...	245,333	$3.27	$0.16	$2.44	$1.12	$6.99
1904 ...	202,286	3.42	0.17	2.67	1.05	7.31
1905 ...	260,433	3.31	0.19	2.45	0.79	6.74
1906 ...	246,593	3.51	0.20	2.06	1.25	7.02
1907 ...	331,629	3.79	0.21	2.27	0.85	7.22

BUTTE COALITION

1906 ...	149,101	$3.94	$0.60	$3.94	$1.50	$9.98
1907 ...	412,169	5.49	0.29	2.29		

[1] The following results have been calculated by James Ralph Finlay from the reports of the respective companies. See *Eng. and Mng. Journal* of April 25 and July 25, 1908.

NORTH BUTTE

1906 ...	259,650	$4.47	$0.20	$4.84		$9.51
1907 ...	374,632	4.53	0.20	4.04		8.77

It is to be noted that the Butte and Boston ores have cost less than the others. This is undoubtedly due to their lower grade, the proportionate cost for smelting, refining, and marketing being less. On the other hand, the North Butte has cost more on account of its higher grade and Butte Coalition has cost more than the average on account of the large expenditures for improvements.

The Tennessee Copper Company employs pyrite smelting to

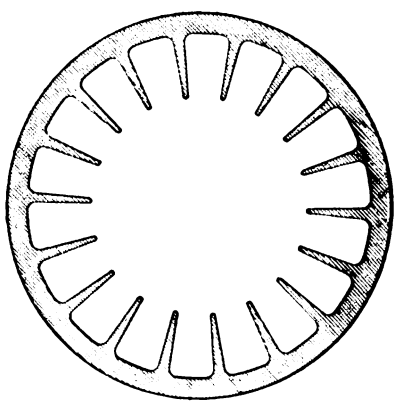


FIG. 121. — Cross-section of U-pipe

work the cupriferous pyrrhotite extracted from large massive lenses. Finlay gives the following results of the year 1907, during which the costs were somewhat higher than normal, due to unsatisfactory economic conditions throughout the country. Since that time they have decreased, and the situation is also improved by the addition of an acid plant which is utilizing successfully the sulphur gases from the blast-furnaces.

COSTS PER TON

Development	$0.1318	
Mining, hoisting, etc.	0.9189	
Crushing and sorting	0.0804	
General	0.0851	
Total current cost	$1.2162	$1.2162

Add cost of preliminary development amortized in 15 years at 5 per cent interest and 5 per cent annual amortization		0.06
Mining plant similarly amortized		0.15
Transportation to smelter	$0.1329	
Blast-furnace	1.6219	
Engineering and laboratory	0.0628	
General	0.0852	
Converting	0.2402	
Total current smelting cost		$2.1430
Add amortization of smelting plant and railway as above		0.47
Add administration, shipping, refining, and selling expenses		0.49
Grand total		$4.5292

On the basis thus figured, anything received above 12c. a pound for copper in New York is applicable to dividends.

The following figures for the year 1909 have been computed from the reports of the respective companies by the *Engineering and Mining Journal*, except in the cases of Calumet and Hecla, United Verde, and Granby, which are estimated. In all cases the proceeds of gold and silver are deducted from the total expense before determining the cost of the copper, except for the U. S. Smelting Co., for which the cost is distributed among the metals produced.

Company	Production	Cost per lb.
Nevada Consolidated	34,527,823	7.14
Baltic	17,814,836	7.98
Champion	18,005,071	8.45
Utah Copper Co.	51,749,233	8.74
Red Metal	34,654,971	8.98
Calumet and Hecla	75,000,000	9.00
United Verde	36,694,063	9.00
Butte and Boston	20,955,910	9.24
Boston and Montana	101,951,350	9.38
Osceola	25,296,657	9.47
Superior and Pittsburg	24,325,667	9.66
Calumet and Arizona	27,630,050	9.68
British Columbia	6,325,000	9.77
North Butte	33,102,153	9.78
Quincy	22,511,984	9.98
Phelps, Dodge, and Co.	108,542,964	10.00
Total < 10c	639,087,732	—

Utah Consolidated	10,043,900	10.07
U. S. Smelting Co.	36,672,606	10.12
Granby	22,000,000	10.50
Trenton	7,168,318	10.61
Tennessee	14,058,954	10.68
Total 10–11	89,943,778	—
Total < 11c	729,031,510	—
Anaconda	76,335,433	11.07
Mohawk	11,248,474	11.21
Arizona	31,962,000	11.41
Old Dominion	25,417,712	11.43
Cananea	44,547,689	11.64
Imperial	10,500,000	11.87
Total 11–12c	200,011,308	—
Total < 12c	929,042,818	—
Shannon	17,553,213	12.79
Franklin	1,615,556	13.35
Allouez	4,031,532	13.39
Trimountain	5,282,404	13.89
Tamarack	13,533,207	14.30
Ahmeek	9,198,110	15.48
Centennial	2,583,793	15.61
Parrot	5,407,255	16.28
Isle Royale	5,719,015	16.64
Victoria	1,062,218	17.09
Total 12–18c	65,986,303	—

Owing to the economic necessity of conducting copper smelting upon a large scale and with the aid of highly-paid scientific men, the majority of copper miners sell their ore to some one of the great central smelting plants. If the copper contained in the ore were the sole object of consideration in such a sale, the transaction would be comparatively simple. It might, probably, consist only of two factors:

1. A fixed charge per ton of the ore — called the "smelting charge."

2. A certain price per pound of copper contained in the ore, which would vary with the market price of the metal as established by the actual sales from day to day in the great commercial centers, such as New York or London.

But even this suppositious elementary case is not so simple as it appears at first reading. A brief study in detail of the two preceding factors will show at once that each of them is a composite proposition made up of a number of uncertain, complicated

and variable elements, and demanding a consideration on the part of the ore-vender that he is seldom either willing or able to devote to the subject.

1. A fixed charge per ton of ore — called the "smelting charge." This charge is made up of two principal elements:

a The actual average cost of smelting a ton of ore, determined by results at that particular plant.

b The net profit on the ton of ore — applicable to dividends or surplus.

The constant aim of the smelter is, of course, to keep (*a*) as small as possible while increasing (*b*) to the highest attainable limit.

The sum (*a*), whether large or small, is an exceedingly complex factor, made up of many and variable elements. It includes not only the ordinary costs of labor, fuel, supplies, salaries, superintendence, office expenses at mine and at headquarters, insurance, taxes, and similar obvious expenses, but also such charges as new construction, in so far as necessary to keep the plant abreast of the times; of amortization of the plant within 15 to 18 years; of grants to employees for accidents and injuries; of damages to crops; of legal costs in fighting blackmailing suits for similar damages; of interest on the large capital tied up in ores apart from the actual cost of the plant; and of many other items which are as real and as inevitable as the cost of coke or labor. All of these elements combine to make the actual cost of smelting a ton of ore far higher than is popularly believed by the miner who is selling it.

It must be recollected also that I am assuming a condition of almost unheard-of simplicity, where the copper in the ore is the sole object of consideration, and where, consequently, the gangue-rock is of so favorable a nature that no complications or undue expenses arise in the smelting process.

(*b*) The net profit which the management has decided to make on each ton of ore smelted. This is a commercial, rather than technical, factor, and needs therefore little consideration in the present discussion. Like every other business man, the smelter desires to have his net profit as large as possible. In this he is limited by competition, as well as by the fact that most mines work on a narrow margin of profit, and would cease to produce if the smelting charges became unduly high. In many instances

it is extremely small, and it is doubtful if, at the present time, the average net profit of the custom smelters of Colorado reaches one dollar per ton of ore treated.

This completes the consideration of the first portion of the smelter's bargain with the miner in purchasing the simple, self-fluxing copper ore which has been taken for illustration. The second portion of the contract is:

2. A certain price per pound copper contained in the ore, and which varies with the market price of that metal. This matter of the price paid per pound for the copper in the ore is a sore subject with most miners, owing to the heavy deductions made from the selling price of the refined metal in New York. With few exceptions, the basis upon which this deduction is made is not generally understood by mining men and, while simple explanation may not result in reconciling them to the price they receive for their copper, it can at least show them that it is not the smelter who is getting the benefit of this deduction.

In the first place, the smelter does not pay for all of the copper shown in the ore by assay, but subtracts a certain fixed amount (usually 1.3 or 1.5) from the copper assay as returned by the chemist. This is an ancient and — in my opinion — objectionable practice among custom smelters, coming down from the time when Swansea controlled much of the copper business of the world. Copper assays at Swansea were made by the dry, or fire, method, and gave results lower than the wet, or chemical, assay, and were supposed to represent about what would actually be obtained when the ores were smelted commercially.

In other words, the Swansea "dry assay" discounted the actual smelting losses, so that the Swansea smelter did not have to make any further deduction for the inevitable losses that accompany the chain of smelting operations, but paid so much per pound of copper for all of the copper in the ore as shown by this dry assay.

As the copper industry expanded in the United States, it was found that this dry assay was too uncertain and too dependent upon the skill of the individual assayer for large operations, and it was given up entirely in favor of the absolutely correct wet, or chemical, assay, and thus made no allowance for metallurgical losses. These losses, however, are inevitable and must be pro-

vided for in some way or another. It thus became a question whether to provide for them by charging a higher price per ton for smelting the ore, or by making a fixed arbitrary deduction from the wet assay which should be sufficient to cover the metallurgical losses. The latter system was adopted, and the arbitrary figure 1.3 (sometimes 1.5) is deducted from the wet assay of the ore, the result being called the "dry assay," and representing the amount of copper that is paid for by the smelter. Thus, if

$$x = \text{wet assay}$$
$$x - 1.3 = \text{dry assay}$$

or, expressing the same in figures, if an ore contains 10 per cent copper by wet assay, its dry assay will be $10 - 1.3 = 8.7$ per cent.

On favorable ores below six per cent copper, it is probable that this deduction of 1.3 is in favor of the smelter. On more difficult ores, with a higher tenor in copper, it is likely to be the reverse. On the whole, it seems to be reasonably fair for both sides, but is an awkward and unfortunate method of conducting commercial transactions.

Having established the assay value in copper upon which payment is to be made for an ore, the smelter proceeds next to make a deduction of from three to five cents, or more, per pound of copper from the actual selling price of that metal in New York. This deduction is so large that it always excites the suspicion and indignation of the miner unless he is thoroughly familiar with the reasons upon which it is based. A consideration of the steps through which the copper has to pass, from the day on which it is received in the ore by the smelter until the much later date when it is sold to the consumer as refined metal, will at least dispel the idea that the smelter obtains any considerable benefit from this deduction.

The principal items of cost to the smelter during the long chain of operations that connects the crude ore as purchased from the miner with the refined copper as sold to the consumer are:

1. The cost of sampling and smelting the ore. This is already provided for under the fixed sum called the "smelting charge."

2. The loss of copper during the smelting, converting, elec-

trolytic refining, furnace refining, etc. This item is also provided for by means of the arbitrary deduction of 1.3 from the wet assay of the ore.

We may assume that the ore has now undergone the first smelting operation, and the copper, minus metallurgical losses, is present in the form of matte — a substance which may be regarded simply as a more or less concentrated sulphide ore free from gangue-rock. Indeed, the conditions are much the same as though the miner owned a concentrating mill, had passed his ore through it, and had sold his sulphide concentrates to the smelters. The matte represents these sulphide concentrates, and the smelter now takes a new departure and submits this matte to a long and expensive series of operations, the cost of which per pound of copper is pretty nearly proportionate to the amount of copper present. His original smelting tariff was only framed to cover the first smelting of the ore into matte, and he now has to face a new series of expenses in extracting the copper from this matte, shipping it to a commercial center, refining it, and paying an agent to sell it. Under ordinary conditions, nothing that he has yet received from the miner is applicable to these costs, with their accompanying metallurgical losses, excepting the 1.3 deduction from the wet assay of the original ore, which is supposed to be sufficient to cover the losses in copper all the way through.

We may now consider the actual cost to the smelter of the operations just enumerated, basing them upon an average of several moderate-sized representative western gold-silver-copper plants.[1]

3. The cost of concentrating the matte to a point fit for converting; of converting the matte; of shipping the converter bars to an eastern refinery; of casting the bars into anodes; of refining the anodes by the electrolytic method; of refining the resulting

[1] While we are not yet considering the question of how gold and silver are paid for as they occur in copper ores, it is necessary to use for our illustration ores that contain the precious metals, for the simple reason that the heavy deduction from the selling price of the copper is made on just this class of ores. Straight copper ores that can be smelted into a convertible matte at a single fusion are generally purchased on an entirely different basis. The former class of ore is the common one, and also the one that usually causes so much dissatisfaction on the part of the miner.

cathode copper into merchantable shapes; of selling the refined copper.

(*a*) Cost of concentrating the matte to a point fit for converting. This is an almost invariable necessity in a custom plant purchasing precious-metal ores and using copper matte as a collector. It is really, for the most part, a gold and silver smelter, using copper incidentally as a collector and, as such, might scarcely seem warranted in claiming so much attention in a work on copper; but, as it is exactly this class of cases which is least understood by the miners and which, consequently, causes the most trouble, it seems worth while to devote some care to its explanation. In the kind of smelting to which I am now referring, a large proportion of the gold and silver ores contain no copper at all; hence, the average percentage of copper in the furnace-mixture is low, and even a high rate of concentration fails to produce a first matte that is high enough in copper to warrant direct bessemerizing. A concentration smelting of this matte is necessary to yield a product fit for converting and, owing to the small amount of copper in the matte, the cost of this matte-concentration process — usually conducted on a small scale — can seldom be put at less than two cents per pound of copper, and more frequently exceeds this considerably.

(*b*) The bessemerizing of this concentrated matte, including the resmelting of slags, and concomitant expenses, may be put at a minimum of 0.75 cents per pound copper at a moderate sized plant.

(*c*) The freight on the converter bars to the Atlantic seaboard varies, of course, with the situation of the smelter and the magnitude of its shipments. Including costs of sampling, loading, and several other small expenses, it may be called one cent per pound copper.

(*d*) The charge at the eastern refinery for casting the bars into anodes, refining the anodes by the electrolytic process, refining the cathode copper into merchantable shapes, may also be put at one cent per pound copper.

(*e*) Shipping the refined copper to market and selling it may be estimated at 0.3 cents per pound copper.

Assembling these details of cost, we find that the expense from first matte to final market, based on the copper present, may figure out as follows:

(*a*) matte concentration	2.0	cts.	per	lb.	Cu.
(*b*) converting	0.75	"	"	"	"
(*c*) freight on converter bars	1.0	"	"	"	"
(*d*) refining	1.0	"	"	"	"
(*e*) shipping and selling refined copper	0.3	"	"	"	"
total per pound copper	5.05	"	"	"	"

It is clear, therefore, that the ordinary deduction of four or five cents per pound copper in the ore is seldom of any financial benefit to the smelter.

The payment for the precious metals in ores is usually so simple and so familiar as to demand but scant notice.

Gold is usually paid for at the rate of $20 per ounce, no deduction being made for loss. While the gold losses during the smelting processes are small, they are by no means zero. So far as investigation has gone, they probably range between two and three per cent on favorable ores. This loss, however, is usually balanced — and often exceeded — by small accessions from ores which contain so little gold that it is not paid for nor counted in the assays. Consequently, it is not uncommon for the smelter to find that the balance sheet of his gold shows a small amount on the plus side.

Silver, being prone to volatilization and slag-losses, is treated accordingly, a deduction of five to ten per cent being made, and the remainder being paid for at New York quotations.

It will be noted that, of all the various deductions and charges that have thus far been made against the ore, the only one that benefits the smelter pecuniarily is the fixed smelting price per ton of ore.

Thus far we have been considering only the valuable metals that the ore may contain, and which are to be produced and sold. We come now to another class of substances which are as important to the technical success of the process as the copper, gold, and silver are to its commercial success. I refer, of course, to the slag-forming constituents of the ore, as well as to certain other substances that tend to cause metal losses, or to injure the quality of the product.

As quartz is the most common gangue-rock of ore deposits, and as the metallurgist cannot exceed a certain limit of silica in his slag, it follows that he is usually overloaded with

excess silica which demands the addition of basic flux, usually in the form of barren limestone. This entails loss in furnace capacity and a heavy additional smelting cost, and the fairest and simplest method of assessing this extra expense is to penalize the silica by charging a certain extra sum — often 10 cents per unit — against the SiO_2 which the ore contains.

The lime, iron, manganese, etc., that the ore contains are favorable constituents and are often paid for at a certain agreed price per unit; as it requires, for instance, approximately two units of CaO to flux one unit of SiO_2, the price is usually arranged in this proportion.

Zinc, being peculiarly obnoxious and damaging in a variety of ways, is often penalized when it exceeds a certain percentage. For instance: it is common to charge 25 cents per unit for any zinc in excess of eight or ten per cent.

Arsenic and antimony make trouble all through the smelting and refining operations, and are often penalized at 25 cents per unit, when present in proportions greater than two or three per cent.[1]

Measures, Values, Weights

The standard of value in the United States is the dollar, divided into 100 cents, and written decimally. Thus $23.37 would signify twenty-three dollars and thirty-seven cents.

One dollar = four shillings one and one-third pence, English; 4.20 marks, German; 5.18 francs, French.

One pound (avoirdupois)	=	0.4536 kilograms
One kilogram	=	2.204 pounds (av.)
One ton (short ton of 2000 lb.)	=	907.2 kilograms
One metric ton	=	2204.6 pounds (av.)
One foot (12 inches)	=	0.3048 meter
One meter	=	3.28 feet
One inch	=	0.0254 meter
One millimeter	=	0.03937 inches
Once ounce per ton (gold or silver in ore)	=	0.0034286 per cent
One per cent. (gold or silver in ore)	=	291.66 oz. per ton
One gallon	=	3.785 liters

[1] See "Principles of Copper Smelting," Chapter XV, for more detailed information regarding the buying and selling of ores.

ATOMIC WEIGHTS [1]

(As used in this book)

Element	Chemical Symbol	Atomic Weight
Aluminum	Al	27.0
Antimony	Sb	120.0
Arsenic	As	75.0
Barium	Ba	137.0
Bismuth	Bi	208.0
Calcium	Ca	40.0
Carbon	C	12.0
Copper	Cu	63.6
Hydrogen	H	1.0
Iron	Fe	56.0
Lead	Pb	207.0
Magnesium	Mg	24.0
Manganese	Mn	55.0
Mercury	Hg	200.0
Nitrogen	N	14.0
Oxygen	O	16.0
Potassium	K	39.0
Silicon	Si	28.0
Silver	Ag	108.0
Sodium	Na	23.0
Sulphur	S	32.0
Zinc	Zn	65.4

The valences of the common elements, expressed by their symbols, are:

Univalent: H, Cl, Br, I, F; Li, Na, K, Ag.

Bivalent: O, S, Se, Te; Mg, Ca, Sr, Ba, Pb, Hg, Cu, Zn, Co, Ni.

Trivalent: B, Au; probably also Al, Fe, Mn, Cr.

Tetravalent: C, Si, Ti, Sn.

Pentavalent: N, P, As, Sb, V, Bi.

Some of the elements show a different valence in different compounds.

By permission of the author, Dr. Leonard Waldo, of New York, I append his thermometric conversion table. This table is exceedingly convenient, and has the merit of exactness.

[1] Most of the weights given in this table correspond to recent determinations; but, in a few instances, minute fractions have been omitted, as ordinary metallurgical calculations require no such extreme accuracy.

Centigrade and Fahrenheit Temperature Conversion Table

C°	0	10	20	30	40	50	60	70	80	90
	F	F	F	F	F	F	F	F	F	F
—200	—328	—346	—364	—382	—400	—418	—436	—454		
—100	—148	—166	—184	—202	—220	—238	—256	—274	—292	—310
—0	+32	+14	—4	—22	—40	—58	—76	—94	—112	—130
0	32	50	68	86	104	122	140	158	176	194
100	212	230	248	266	284	302	320	338	356	374
200	392	410	428	446	464	482	500	518	536	554
300	572	590	608	626	644	662	680	698	716	734
400	752	770	788	806	824	842	860	878	896	914
500	932	950	968	986	1004	1022	1040	1058	1076	1094
600	1112	1130	1148	1166	1184	1202	1220	1238	1256	1274
700	1292	1310	1328	1346	1364	1382	1400	1418	1436	1454
800	1472	1490	1508	1526	1544	1562	1580	1598	1616	1634
900	1652	1670	1688	1706	1724	1742	1760	1778	1796	1814
1000	1832	1850	1868	1886	1904	1922	1940	1958	1976	1994
1100	2012	2030	2048	2066	2084	2102	2120	2138	2156	2174
1200	2192	2210	2228	2246	2264	2282	2300	2318	2336	2354
1300	2372	2390	2408	2426	2444	2462	2480	2498	2516	2534
1400	2552	2570	2588	2606	2624	2642	2660	2678	2696	2714
1500	2732	2750	2768	2786	2804	2822	2840	2858	2876	2894
1600	2912	2930	2948	2966	2984	3002	3020	3038	3056	3074
1700	3092	3110	3128	3146	3164	3182	3200	3218	3236	3254
1800	3272	3290	3308	3326	3344	3362	3380	3398	3416	3434
1900	3452	3470	3488	3506	3524	3542	3560	3578	3596	3614
2000	3632	3650	3668	3686	3704	3722	3740	3758	3776	3794
2100	3812	3830	3848	3866	3884	3902	3920	3938	3956	3974
2200	3992	4010	4028	4046	4064	4082	4100	4118	4136	4154
2300	4172	4190	4208	4226	4244	4262	4280	4298	4316	4334
2400	4352	4370	4388	4406	4424	4442	4460	4478	4496	4514
2500	4532	4550	4568	4586	4604	4622	4640	4658	4676	4694
2600	4712	4730	4748	4766	4784	4802	4820	4838	4856	4874
2700	4892	4910	4928	4946	4964	4982	5000	5018	5036	5054
2800	5072	5090	5108	5126	5144	5162	5180	5198	5216	5234
2900	5252	5270	5288	5306	5324	5342	5360	5378	5396	5414
3000	5432	5450	5468	5486	5504	5522	5540	5558	5576	5594
3100	5612	5630	5648	5666	5684	5702	5720	5738	5756	5774
3200	5792	5810	5828	5846	5864	5882	5900	5918	5936	5954
3300	5972	5990	6008	6026	6044	6062	6080	6098	6116	6134
3400	6152	6170	6188	6206	6224	6242	6260	6278	6296	6314
3500	6332	6350	6368	6386	6404	6422	6440	6458	6476	6494
3600	6512	5630	6548	6566	6584	6602	6620	6638	6656	6674
3700	6692	6710	6728	6746	6764	6782	6800	6818	6836	6854
3800	6872	6890	6908	6926	6944	6962	6980	6998	7016	7034
3900	7052	7070	7088	7106	7124	7142	7160	7178	7196	7214
C°	0	10	20	30	40	50	60	70	80	90

C°	F°
1	1.8
2	3.6
3	5.4
4	7.2
5	9.0
6	10.8
7	12.6
8	14.4
9	16.2
10	18.0

F°	C°
1	.56
2	1.11
3	1.67
4	2.22
5	2.78
6	3.33
7	3.89
8	4.44
9	5.00
10	5.56
11	6.11
12	6.67
13	7.22
14	7.78
15	8.33
16	8.89
17	9.44
18	10.00

Examples: 1347°C = 2444°F + 12°.6F = 2456°.6F:
3367°F = 1850°C + 2°.78C = 1852°.78C

square meters	×	10.76	= square feet
square feet	×	0.093	= square meters
cubic meters	×	35.315	= cubic feet
cubic feet	×	0.028317	= cubic meters
kg. per square meter	×	0.2049	= lb. per square foot
lb. per square foot	×	4.88	= kg. per square meter
met. tons per square meter	×	0.10246	= short tons per square foot
short tons per square foot	×	9.76	= met. tons per square meter

1 lb. Fe	makes 1.286	lb. FeO	
1 lb. Fe	makes 1.43	lb. Fe_2O_3	
1 lb. Fe	makes 1.571	lb. FeS	
1 lb. FeO	makes 1.113	lb. Fe_2O_3	
1 lb. FeO	makes 1.222	lb. FeS	
1 lb. Fe_2O_3	makes 0.9	lb. FeO	
1 lb. FeS	makes 0.818	lb. FeO	
1 lb. FeS_2	makes 0.6	lb. FeO	
1 lb. FeS_2	makes 0.733	lb. FeS	
1 lb. FeS	requires 0.5456	lb. O	to burn to FeO + SO_2
1 lb. O	burns 1.833	lb. FeS	to FeO + SO_2
1 lb. FeS_2	requires 0.4	lb. O	to burn to FeO + SO_2 + S
1 lb. O	burns 2.5	lb. FeS_2	to FeO + SO_2 + S
1 lb. S	makes 2.75	lb. FeS	

Hearth Area of Blast-Furnaces (sq. Ft.)

Width (or Diam.)	Area of a Round Furnace	Length in Inches													
		1	56	60	64	66	72	78	80	84	90	96	100	108	120
24	3.14	.166	9.33	10.00	10.66	11.00	12.00	13.00	13.33	14.00	15.00	16.00	16.67	18.00	20.00
26	3.69	.181	10.11	10.83	11.55	11.91	13.00	14.08	14.44	15.16	16.25	17.33	18.06	19.50	21.67
28	4.28	.194	10.89	11.67	12.44	12.83	14.00	15.16	15.55	16.33	17.50	18.67	19.44	21.00	23.33
30	4.91	.208	11.67	12.50	13.33	13.75	15.00	16.25	16.66	17.50	18.75	20.00	20.83	22.50	25.00
32	5.59	.222	12.44	13.33	14.22	14.66	16.00	17.33	17.77	18.66	20.00	21.33	22.22	24.00	26.66
34	6.30	.236	13.22	14.16	15.11	15.58	17.00	18.41	18.88	19.83	21.25	22.67	23.61	25.50	28.33
36	7.07	.250	14.00	15.00	16.00	16.50	18.00	19.50	20.00	21.00	22.50	24.00	25.00	27.00	30.00
38	7.88	.264	14.73	15.88	16.95	17.42	19.00	20.58	21.11	22.17	23.75	25.33	26.39	28.50	31.67
40	8.73	.278	15.56	16.67	17.89	18.33	20.00	21.67	22.22	23.33	25.00	26.66	27.77	30.00	33.33
42	9.62	.292	16.33	17.50	18.75	19.25	21.00	22.75	23.33	24.50	26.25	28.00	29.17	31.50	35.00
44	10.56	.306	17.12	18.33	19.61	20.16	22.00	23.83	24.44	25.67	27.50	29.33	30.55	33.00	36.67
46	11.54	.320	17.94	19.17	20.50	21.08	23.00	24.91	25.55	26.83	28.75	30.67	31.95	34.50	38.33
48	12.57	.333	18.67	20.00	21.33	22.00	24.00	26.00	26.66	28.00	30.00	32.00	33.33	36.00	40.00
50	13.64	.347	19.44	20.83	22.22	22.91	25.00	27.08	27.77	29.17	31.25	33.33	34.71	37.50	41.67
52	14.75	.361	20.22	21.67	23.10	23.83	26.00	28.16	28.88	30.33	32.50	34.67	36.10	39.00	43.33
54	15.90	.375	21.00	22.50	24.00	24.75	27.00	29.25	29.99	31.50	33.75	36.00	37.49	40.50	45.00
56	17.10	.389	21.78	23.33	24.88	25.66	28.00	30.32	31.11	32.66	35.00	37.33	38.88	42.00	46.66

The areas of many furnaces not shown can readily be ascertained, thus: a 33″ x 96″ furnace area is ½ the sum of a 32″ and 34″ of same length, a 32″ x 54″ is ½ of a 32″ x 108″ and a 42″ x 144″ is double that of a 42″ x 72″.[1]

[1] Calculated by Mr. H. B. Lowden.

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